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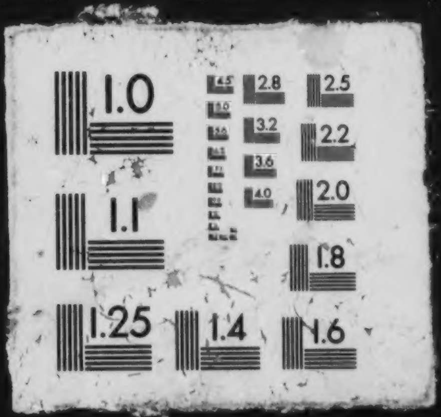
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LEACHABILITY AND REVEGETATION OF SOLID WASTE FROM MINING

by

M. Lynn Apel
Energy Pollution Control Diviion
Industrial Environmental Research Laboratory
Cincinnati, Ohio 45268

INDUSTRIAL ENVIRONMENTAL RESEARCH LABORATORY
OFFICE OF RESEARCH AND DEVELOPMENT
U.S. ENVIRONMENTAL PROTECTION AGENCY
CINCINNATI, OHIO 45268

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This research study was conducted to assess the effectiveness in pollution abatement of various mining solid waste disposal strategies. Column studies were undertaken to evaluate the quality and quantity of leachate generated by the disposal of a pyrite mine waste under various soil amelioration and layering configurations and to assess the vegetative uptake of potentially hazardous materials from the solid waste.

Columns containing the mine waste under one to four feet of cover soil were used to assess the capability of the cover material in reducing leachate volume, improving leachate quality, and enhancing vegetative growth. Concurrently, columns containing neutralizing materials were used to determine if such materials aided in retarding metal and acid migration throughout the soil.

The results of this study illustrated that the quality of leachate resulting from the disposal of a pyrite mine waste may be dramatically improved by incorporating lime, sewage sludge, and fertilizer into the upper strata of the mine waste, layering sewage sludge and fertilizer on top of the lime-treated mine waste, or by covering the mine waste with a relatively heavy layer of cover soil.

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FOREWORD

When energy and material resources are extracted, processed, converted, and used, the related pollutional impacts on our environment and even on our health often require that new and increasingly more efficient pollution control methods be used. The Industrial Environmental Research Laboratory-Cincinnati (IERL-Ci) assists in developing and demonstrating new and improved methodologies that will meet these needs both efficiently and economically.

This report describes a series of pilot plant column studies performed to assess the leachability and revegetative characteristics of a pyrite mine waste. Various layering configurations were evaluated utilizing the mine waste, a cover soil material, and soil amendments. These configurations were compared with respect to mine drainage abatement and vegetative metal uptake. This report will be of interest to persons in the mining industry and government regulatory agencies concerned with the reclamation of mined land or acidic solid waste materials. In addition, it may be of interest to researchers designing pilot plant column studies. For further information contact the Nonferrous Metals and Mineral Mining Branch, IERL-Cincinnati.

David G. Stephan
Director
Industrial Environmental Research Laboratory
Cincinnati

ABSTRACT

This research study was conducted to assess the effectiveness in pollution abatement of various mining solid waste disposal strategies. Column studies were undertaken to evaluate the quality and quantity of leachate generated by the disposal of a pyrite mine waste under various soil amelioration and layering configurations and to assess the vegetative uptake of potentially hazardous materials from the solid waste.

Columns containing the mine waste under 0.3 to 1.2 meters (one to four feet) of cover soil were used to assess the capability of the cover material to reduce leachate volume, improve leachate quality, and enhance the growth of cover vegetation. Concurrently, columns containing neutralizing materials were used to determine if such materials aided in retarding metal and acid migration throughout the soil.

The results of this study illustrate that the quality of leachate resulting from the disposal of pyrite mine waste may be dramatically improved by incorporating lime, sewage sludge, and fertilizer into the upper strata of the mine waste; layering sewage sludge and fertilizer on top of the lime-treated mine waste; or by covering the mine waste with a relatively thick layer of cover soil.

This report covers the period from October, 1979 to September, 1981, and work was completed as of November, 1981.

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CONTENTS

Foreward iii

Abstract iv

Figures vi

Tables vii

Acknowledgments viii

1. Introduction 1

2. Conclusion and Recommendations 3

3. Background 5

    The geochemistry of acid mine drainage 5
    Contrary Creek 7
    Utilization of municipal sludge in other mine reclamation efforts 13

4. Materials and Methods 16

    Climatological conditions 16
    Engineering design 21
    Experimental design 24
    Methods used for data analysis and characterization 30

5. Results 32

    Mine waste and cover soil analyses 32
    Analysis and characterization of the leachate 32
    Column leachate pH and acidity data 34
    Column leachate specific conductance and metal data 63
    Assessment of metal migration throughout the mine waste and soil strata 127
    Vegetative metal uptake determinations 128

6. Discussion 130

References 136

BEGIN LAST LINE OF TEXT
BOTTOM OF IMAGE AREA; OUTSIDE DIMENSION FOR TABLES AND ILLUSTRATIONS

PAGE NUMBER
EPA Form 2360-3 (2-80)
TYPING GUIDE SHEET

CONTENTS (continued)

Appendices 139

A. Column leachate flow data 139
B. Column leachate physical and inorganic data 144
C. Column leachate metal concentration data 149
D. Vegetative metal concentration data 154
E. Soil and mine waste metal migration data 156

LIST OF FIGURES

Figure No. Page No.

1. Location of Contrary Creek Project Area 8
2. Map of Contrary Creek Project Site 9
3. Location of Test Columns in the Greenhouse of the Test and Evaluation Facility 20
4. Column Design 22
5. Internal Leachate Collection System 23
6. Contents of Columns 1 through 17 29
7. Particle Size Distribution of the Mine Waste 33
8. Graph of pH as a Function of Time for Column 1 35
9. Graph of pH as a Function of Time for Column 2 36
10. Graph of pH as a Function of Time for Column 3 37
11. Graph of pH as a Function of Time for Column 4 38
12. Graph of pH as a Function of Time for Column 5 39
13. Graph of pH as a Function of Time for Column 6 40
14. Graph of pH as a Function of Time for Column 7 41
15. Graph of pH as a Function of Time for Column 8 42
16. Graph of pH as a Function of Time for Column 9 43
17. Graph of pH as a Function of Time for Column 10 44
18. Graph of pH as a Function of Time for Column 11 45
19. Graph of pH as a Function of Time for Column 12 46
20. Graph of pH as a Function of Time for Column 13 47

21. Graph of pH as a Function of Time for Column 14 48

22. Graph of pH as a Function of Time for Column 15 49

23. Graph of pH as a Function of Time for Column 16 50

24. Graph of pH as a Function of Time for Column 17 51

25. Graph of Mean Acidity Load as a Function of Time for Columns 1 and 2 53

26. Graph of Mean Acidity Load as a Function of Time for Columns 3 and 4 54

27. Graph of Mean Acidity Load as a Function of Time for Columns 5 and 6 55

28. Graph of Mean Acidity Load as a Function of Time for Columns 7 and 8 56

29. Graph of Mean Acidity Load as a Function of Time for Columns 9 and 10 57

30. Graph of Mean Acidity Load as a Function of Time for Columns 11 and 12 58

31. Graph of Mean Acidity Load as a Function of Time for Columns 13 and 14 59

32. Graph of Acidity Load as a Function of Time for Column 15 60

33. Graph of Acidity Load as a Function of Time for Column 16 61

34. Graph of Acidity Load as a Function of Time for Column 17 62

35. Graph of Specific Conductance as a Function of Time for Column 1 64

36. Graph of Specific Conductance as a Function of Time for Column 2 65

37. Graph of Specific Conductance as a Function of Time for Column 3 66

38. Graph of Specific Conductance as a Function of Time for Column 4 67

39. Graph of Specific Conductance as a Function of Time for Column 5 68

40. Graph of Specific Conductance as a Function of Time for Column 6 69

41. Graph of Specific Conductance as a Function of Time for Column 7 70

42. Graph of Specific Conductance as a Function of Time for Column 8 71

43. Graph of Specific Conductance as a Function of Time for Column 9 72

44. Graph of Specific Conductance as a Function of Time for Column 10 73

45. Graph of Specific Conductance as a Function of Time for Column 11 74

46. Graph of Specific Conductance as a Function of Time for Column 12 75

47. Graph of Specific Conductance as a Function of Time for Column 13 76

48. Graph of Specific Conductance as a Function of Time for Column 14 77

49. Graph of Specific Conductance as a Function of Time for Column 15 78

50. Graph of Specific Conductance as a Function of Time for Column 16 79

51. Graph of Specific Conductance as a Function of Time for Column 17 80

52. Graph of Mean Iron Load as a Function of Time for Columns 1 and 2 83

53. Graph of Mean Iron Load as a Function of Time for Columns 3 and 4 84

54. Graph of Mean Iron Load as a Function of Time for Columns 5 and 6 85

55. Graph of Mean Iron Load as a Function of Time for Columns 7 and 8 86

56. Graph of Mean Iron Load as a Function of Time for Columns 9 and 10 87

57. Graph of Mean Iron Load as a Function of Time for Columns 11 and 12 88

58. Graph of Mean Iron Load as a Function of Time for Columns 13 and 14 89

59. Graph of Iron Load as a Function of Time for Column 15 90

60. Graph of Iron Load as a Function of Time for Column 16 91

61. Graph of Iron Load as a Function of Time for Column 17 92

62. Graph of Mean Copper Load as a Function of Time for Columns 1 and 2 94

63. Graph of Mean Copper Load as a Function of Time for Columns 3 and 4 95

64. Graph of Mean Copper Load as a Function of Time for Columns 5 and 6 96

65. Graph of Mean Copper Load as a Function of Time for Columns 7 and 8 97

66. Graph of Mean Copper Load as a Function Time for Columns 9 and 10 98

67. Graph of Mean Copper Load as a Function of Time for Columns 11 and 12 99

68. Graph of Mean Copper Load as a Function of Time for Columns 13 and 14 100

69. Graph of Copper Load as a Function of Time for Column 15 101

70. Graph of Copper Load as a Function of Time for Column 16 102

71. Graph of Copper Load as a Function of Time for Column 17 103

72. Graph of Mean Zinc Load as a Function of Time for Columns 1 and 2 106

73. Graph of Mean Zinc Load as a Function of Time for Columns 3 and 4 107

74. Graph of Mean Zinc Load as a Function of Time for Columns 5 and 6 108

75. Graph of Mean Zinc Load as a Function of Time for Columns 7 and 8 109

76. Graph of Mean Zinc Load as a Function of Time for Columns 9 and 10 110

77. Graph of Mean Zinc Load as a Function of Time for Columns 11 and 12 111

78. Graph of Mean Zinc Load as a Function of Time for Columns 13 and 14 112

79. Graph of Zinc Load as a Function of Time for Column 15 113

80. Graph of Zinc Load as a Function of Time for Column 16 114

81. Graph of Zinc Load as a Function of Time for Column 17 115

82. Graph of Mean Lead Load as a Function of Time for Columns 1 and 2 117

83. Graph of Mean Lead Load as a Function of Time for Columns 3 and 4 118

84. Graph of Mean Lead Load as a Function of Time for Columns 5 and 6 119

85. Graph of Mean Lead Load as a Function of Time for Columns 7 and 8 120

86. Graph of Mean Lead Load as a Function of Time for Columns 9 and 10 121

87. Graph of Mean Lead Load as a Function of Time for Columns 11 and 12 122

88. Graph of Mean Lead Load as a Function of Time for Columns 13 and 14 123

89. Graph of Lead Load as a Function of Time for Column 15 124

90. Graph of Lead Load as a Function of Time for Column 16 125

91. Graph of Lead Load as a Function of Time for Column 17 126

92. Combined Metal and Acidity Mean Load Data for Columns 1 and 2 Containing the Untreated Mine Waste 133

93. Combined Metal and Acidity Mean Load Data for Columns 5 and 6 Containing the Lime/Sludge/Fertilizer Incorporated into the Upper Strata of the Mine Waste 133

94. Combined Metal and Acidity Mean Load Data for Columns 7 and 8 Containing a Sludge/Fertilizer Layer over the Lime-Treated Mine Waste 133

95. Combined Metal and Acidity Load Data for Columns 9 and 10 Containing the Mine Waste under 0.46m (1.5 ft) of the Cover Soil 134

96. Combined Metal and Acidity Load Data for Columns 11 and 12 Containing the Mine Waste under 0.61m (2 ft) of Cover Soil 134

97. Combined Metal and Acidity Load Data for Columns 13 and 14 Containing the Mine Waste under 1.2m (4 ft) of Cover Soil 134

EPA-287 (Cin.)
(4-76)

BEGIN FIRST LINE OF TEXT HERE
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TABLES

Number Page

1. EPA Recommended Maximum Trace Metal Loadings For Agricultural Land 14
2. Mean Monthly Precipitation in Centimeters for the Study Area from 1974 through 1978 17
3. Climatological Conditions Simulated in the Greenhouse throughout the Study Period 18
4. Mean Monthly Temperature Data in ^{o}C for the Study Area from 1974 through 1978 19
5. Mining Solid Waste and Cover Soil Analysis 25
6. Column Contents 26
7. Metal and Organic Carbon Analysis of Sludge 27
8. Results of Metal Analysis of the Mine Waste and the Cover Soil Material 82
9. Experimental Column Percent Load Decreases as Compared to the Untreated Mine Waste Control Columns 131

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PAGE NUMBER
EPA Form 2350-4 (4-80)
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[PREVIOUSLY CIN. EPA FORM 257]
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SECTION 1

INTRODUCTION

A vast amount of solid waste is generated through the mining of mineral ores and must be disposed of in an environmentally safe manner. It has been estimated about 1.5 billion tonnes (1.7 billion tons) of mineral wastes are discarded annually in the United States with the total accumulated mineral solid wastes now approaching 23 billion tonnes (25 billion tons) (Dean and Havens, 1973). Mineral waste piles currently cover over 800,000 hectares of land (approximately 2,000,000 acres) and represent over 30 percent of the total wastes produced in the United States (Donovan et al., 1976).

If not properly disposed of, some mineral mine wastes may cause adverse environmental and health related effects. Depending on the ore being mined and the processes used, mine wastes may contain potentially toxic substances including arsenic, cyanide, mercury, and heavy metals. These materials can be subsequently leached from waste piles and pollute nearby groundwater or surface streams. In addition, lack of a vegetative cover over waste piles is conducive to severe wind and water erosion of the materials, resulting in the wind and waterborne transport of potentially toxic materials inherent in the wastes.

Over the past several years the United States government has shown an increasing interest in the proper disposal of solid wastes generated by the mining industry. Congress has passed several pieces of environmental legislation regulating the treatment and disposal of solid wastes representing a danger to public health or the environment. Of recent concern is the Resource Conservation and Recovery Act of 1976, (RCRA, PL 94-580) and it's subsequent amendments. This act is implemented and enforced by the United States Environmental Protection Agency (U.S. EPA) and intended to control the disposal of municipal and industrial solid wastes. The Surface Mining Control and Reclamation Act (SMCRA, PL 95-87), which is implemented and enforced by the Department of Interior, was passed in 1977 and is intended to control the environmental effects of surface coal mining operations and the surface effects of underground coal mining. The government legislation regulating the disposal of uranium mill tailings is the Uranium Mill Tailings Radiation Control Act of 1978 (UMTRCA, PL 95-604), which is also implemented by the U.S. EPA but enforced by the Nuclear Regulatory Commission. These pieces of legislation have contributed to the impetus to develop solid waste disposal techniques that are not only practical and economically feasible but also utilize recycle and reuse technologies whenever possible.

This report presents the results of a series of pilot plant column studies performed to assess the effectiveness in pollution abatement of various mining solid waste disposal strategies. The strategies examined in this study were ones that are currently being used or which have the potential to be used in the disposal of pyritic mine wastes and possibly other acidic solid wastes of similiar character and composition. The objectives of these studies included determination of the physical and chemical quality of leachates generated from test columns containing an acidic mine waste under various soil amelioration and layering configurations, determination of the quantity of leachate produced relative to water input based on rainfall statistics of the geographical location under study, and assessment of the vegetative uptake of potentially hazardous materials from the mining solid waste.

Three approaches were examined for treating the surface of a disposal area in order to mitigate metal and acidity loads in the acid mine drainage resulting from disposal of a pyritic mine waste. The first approach used digested sewage sludge, agricultural limestone, and commercial fertilizer to treat the surface of a waste pile in order to neutralize the acidic waste and resulting drainage and develop a plant growth supporting medium. The second approach utilized a layer of cover soil placed over the mine waste to achieve the same goal. The third approach involved placement of a neutralizing layer between the mine waste and cover soil layer. In all cases the treated surface was planted with vegetation.

This research was conducted in the greenhouse of the U.S. EPA Test and Evaluation Facility (T&E Facility), in Cincinnati, Ohio. It should be noted that the results obtained from these column studies reflect the experimental conditions under which they were obtained and may or may not be indicative of what would occur at an actual mine site during the same period of time.

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TYPING GUIDE SHEET

SECTION 2

CONCLUSIONS AND RECOMMENDATIONS

The results of this research indicate that the quality of leachate emanating from this pyrite mine waste may be notably improved by all of the treatment techniques studied in this experiment, i.e., incorporating lime, municipal sludge, and fertilizer into the upper strata of the mine waste; placing a mixture of sludge and fertilizer on top of the lime- treated mine waste; or by covering the mine waste with a 0.6-m to 1.2-m (2-ft to 4-ft) layer of cover soil. Placement of a soil layer greater than 0.6m (2ft) over the mine waste, however, was found to be the best treatment technique. This technique not only improved leachate quality, but also provided a soil stratum highly suitable for vegetation.

The utilization of municipal wastewater sludge in combination with limestone and fertilizer proved to be highly beneficial in the establishment of a vegetative cover over the mine waste. This pilot plant observation is in agreement with the results obtained at the Contrary Creek field demonstration site. Reclamation schemes including the use of sludge and other amendments, such as those studied here, are recommended for lands containing wastes which exhibit physical and chemical characteristics similar to the mine waste studied in this project. These schemes not only provide a means of sludge disposal but also may have economic advantages depending on the extent and location of the waste material.

Both the layering and incorporation of sludge, lime and fertilizer into the upper portion of the pyrite waste resulted in an improvement in leachate quality comparable to that noted when the mine waste was covered with 1.2m (4ft) of cover soil. Incorporating the amendments, as opposed to layering the amendments, appeared to result in more significant decreases in leachate acidity and metal loads.

Placement of a lime layer between the mine waste and cover soil did not appear to enhance leachate quality during the study period; in fact, leachate quality tended to be worse than that from the control column. This was found to be correlated to a 15-20% decrease in the flow rate for the experimental columns as compared to the control column.

As expected, vegetation was quickly established on the cover soil material placed above the mine waste, and no vegetation was established on the untreated mine waste control columns. Vegetative growth was much slower and about 75% less dense on columns containing the amendments layered and incorporated into the mine waste as compared to the columns containing a cover

EPA Form 2350-4 (4-80)
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TYPING GUIDE SHEET

soil layer over the mine waste. Overall vegetative metal uptake by plants was minimal in all experimental columns.

It was noted that the adverse conditions of the mine waste (i.e. high acidity, low pH, high iron content) were greater deterrents to root penetration into the mine waste than lack of precipitation, as often expected. In addition, a thin layer of lime positioned between the cover soil and the mine waste materials was found to be highly conducive to root pentration into the mine waste layer.

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SECTION 3

BACKGROUND

THE GEOCHEMISTRY OF ACID MINE DRAINAGE

Acid mine drainage is formed upon the exposure of chemically reduced metal sulfides, such as iron disulfides (FeS_2), to atmospheric oxygen and water. The predominant iron disulfide ores are pyrite and marcasite. Pyrite is a hard, brittle, brass-yellow mineral with a metallic luster, commonly known as "fool's gold", while marcasite is a pale bronze-yellow to nearly white mineral. Even though the ratio of sulfur atoms to iron atoms is the same for both minerals, their crystallographic properties are quite different. Marcasite has an orthorhombic structure while pyrite is isometric. In addition, marcasite is less stable and more easily decomposed than pyrite. Pyrite is considered the most widespread of all sulfide minerals and, as a result of it's greater abundance in the eastern United States, pryite is recognized as the major source of acid mine drainage.

The geochemistry of acid mine drainage is summarized in the equations below (Singer and Stumm, 1968). The exposure of pyrite to air and water during mining operations contributes to the oxidation of sulfide to the sulfate ion (equation 1) resulting in the formation of dissolved ferrous sulfate and sulfuric acid.

$$2FeS_2 + 7O_2 + 2H_2O \rightarrow 2Fe^{+2} + 4SO_4^{-2} + 4H^+ \qquad (1)$$

The ferrous iron (Fe^{+2}) is oxidized to ferric iron (Fe^{+3}) (equation 2).

$$4Fe^{+2} + O_2 + 4H^+ \rightarrow 4Fe^{+3} + 2H_2O \qquad (2)$$

The ferric iron then hydrolyzes to form insoluble ferric hydroxide ($Fe(OH)_3$), commonly known as yellow boy, and additional acidity (equation 3).

$$Fe^{+3} + 3H_2O \rightarrow Fe(OH)_3 + 3H^+ \qquad (3)$$

Ferric iron can also be reduced by pyrite itself where sulfide is again oxidized and acidity released along with ferrous iron which may re-enter the cycle (equation 4).

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(PREVIOUSLY CIN: EPA FORM 287)
TYPING GUIDE SHEET

$$FeS_2 + 14Fe^{+3} + 8H_2O \longrightarrow 15Fe^{+2} + 2SO_4^{-2} + 16H^+ \quad (4)$$

The concentration of sulfate or acidity in the water can be directly correlated to the amount of pyrite which has been dissolved. The dissolution of one mole of iron pyrite yields two equivalents of acidity (equation 1), one of which is used in the oxidation of ferrous to ferric iron (equation 2). Equation 3 shows the production of three equivalents of acidity resulting from the hydrolysis of ferric iron. Overall, one mole of pyrite ultimately results in four moles of acidity. Likewise, equation 4 indicates the production of 16 moles of acidity per mole of pyrite. However, since the formation of ferric iron utilizes a hydrogen ion (equation 2), the net production of acidity is two moles per mole of pyrite as in the case of oxidation by oxygen (equation 1).

Singer and Stumm (1970) have concluded that the oxidation of ferrous iron to ferric iron is the rate limiting step in the formation of acid drainage. Above pH 5 this reaction proceeds rapidly. However as the pH decreases, the rate of this reaction also decreases such that at a pH of 3.5 the half-reaction time is on the order of three years.

Numerous parameters have been shown to effect the rate of acid mine drainage formation. Among these are pH, temperature, oxygen concentration, particle size, oxygen transport rate, electron transfer rate, the concentration and type of anions, and the presence and level of certain bacteria. Nelson and Snow (Nelson *et al.*, 1933) observed that the rate of oxidation of pyrite sulfur in coal by oxygen was markedly influenced by the degree of fineness of the coal, the rate being inversely proportionaly to the average diameter of the coal particles. The rate of oxidation has also been found to be dependent upon the amount of oxygen available to the pyrite surface. In addition, it has been noted that the reaction rate doubles for each increase of 10°C (Morth, 1965).

It has also been noted that certain bacteria play a key role in the oxidation of pyrite and the subsequent generation of acid mine drainage. *Thiobacillus ferrooxidans*, an iron oxidizing acidophilic bacterium indigenous to acid drainage, is capable of greatly accelerating the rate of ferrous iron oxidation at low pH's. Under optimum conditions, *T. ferrooxidans* has been shown to increase the rate of this reaction (equation 1) by 5 x 10^5 times (Lacey and Lawson, 1970) with a concomitant increase in the formation of acid drainage.

Data indicate that in addition to directly oxidizing pyrite, acidophilic iron oxidizing bacteria may also be indirectly responsible for pyrite oxidation via the generation of ferric ions which can react to oxidize more pyrite with the concurrent regeneration of ferrous ions (equation 5) (Temple and Kohler, 1954).

$$\begin{tikzcd} 14Fe^{+3} + FeS_2 + 8H_2O \arrow[r] & 15Fe^{+2} + 2[S] \arrow[l, bend left, "\underline{T.\ ferrooxidans}"] \arrow[r, "\underline{\text{Thiobacilli}}", "\text{or } O_2"'] & 16H^+ + 2SO_4^{-2} \end{tikzcd} \quad (5)$$

The regenerated ferrous iron could then be reoxidized by

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TYPING GUIDE SHEET

T. ferrooxidans so that the cycle can continue over and over (Silverman *et al.*, 1961).

CONTRARY CREEK

Location and Mining History

Contrary Creek is located in east central Virginia approximately 65 kilometers (40.3 miles) northwest of Richmond and 120 kilometers (74.4 miles) southwest of Washington, D. C. (Figure 1). The entire creek is located in Louisa County, Va. at 38° 3' N and 77° 53' W near the town of Mineral. The creek is a tributary to a major recreational waterway, Lake Anna, which serves as a reservoir for the North Anna Nuclear Power Plant.

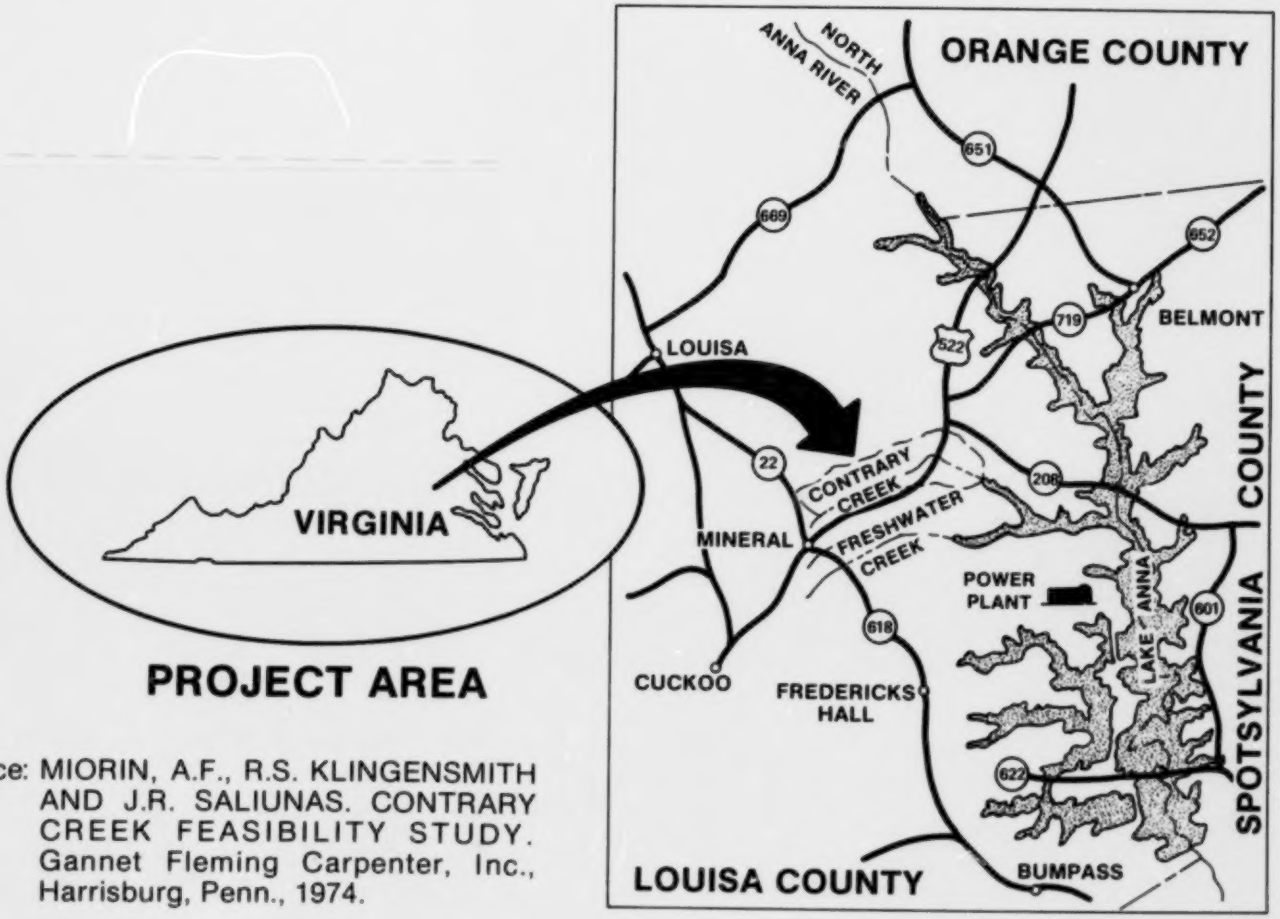
Contrary Creek is approximately eight kilometers (five miles) long and drains an 18 square kilometer area (seven square miles) which includes three abandoned pyrite mines, Arminius, Boyd Smith and Sulphur, and 17 hectares (42 acres) of pyrite tailings that have remained since mining ceased in the early 1920's (Figure 2).

Mining first began at the Arminius mine site in 1834 when gossan iron ore was excavated from surface pits (Hinkle, 1980). During the mid-1800's iron mining was expanded to both the Sulfur and Boyd Smith sites. Copper also was mined from a zone immediately below the iron ore during the same period of time. In the late 1800's pyrite deposits discovered below the iron/copper layers began to be mined and to be used for the manufacture of sulfuric acid. During the period between 1885 and 1920 about 6.3 million tonnes (seven million tons) of pyrite were mined from the three sites via deep mining techniques (Hill *et al.*, 1977). Some of the deep mine shafts in the area exceeded 300 m (91.4 ft) in depth. The pyrite ore from the mines was processed at all three of the mine sites. After being brought to the surface, the ore was crushed and washed with reject material before dumped near each of the mines.

Due to these mining activities, much of the Contrary Creek area became devoid of vegetation with the creek and its tributary streams being severely polluted with acid mine drainage. In the mid 1970's state and federal agencies began an effort to reclaim the Contrary Creek area. These activities resulted in reclamation of approximately 80% of the area (Hinkle, 1980). Today, most of the tailing piles have been regraded and revegetated but areas still exist, particularly along the creek bed, where high acidity and the erosion action of the stream have prevented successful reclamation and acid mine drainage abatement.

Acid Metal Sources at Contrary Creek

Nordstrom and Dagenhart (1979) found that acid seeps from the base of pyritic tailing piles contributed to a major portion of the acid metals entering Contrary Creek during dry periods between storms. Although


ORANGE COUNTY
NORTH ANNA RIVER
651
669
652
719
522
BELMONT
LOUISA
22
CONTRARY CREEK
FRESHWATER CREEK
MINERAL
208
SPOTSYLVANIA COUNTY
LAKE ANNA
POWER PLANT
601
618
622
CUCKOO
FREDERICKS HALL
BUMPASS
LOUISA COUNTY
VIRGINIA
PROJECT AREA

Reference: MIORIN, A.F., R.S. KLINGENSMITH AND J.R. SALIUNAS. CONTRARY CREEK FEASIBILITY STUDY. Gannet Fleming Carpenter, Inc., Harrisburg, Penn., 1974.

Figure 1. Location of Contrary Creek Project Area.

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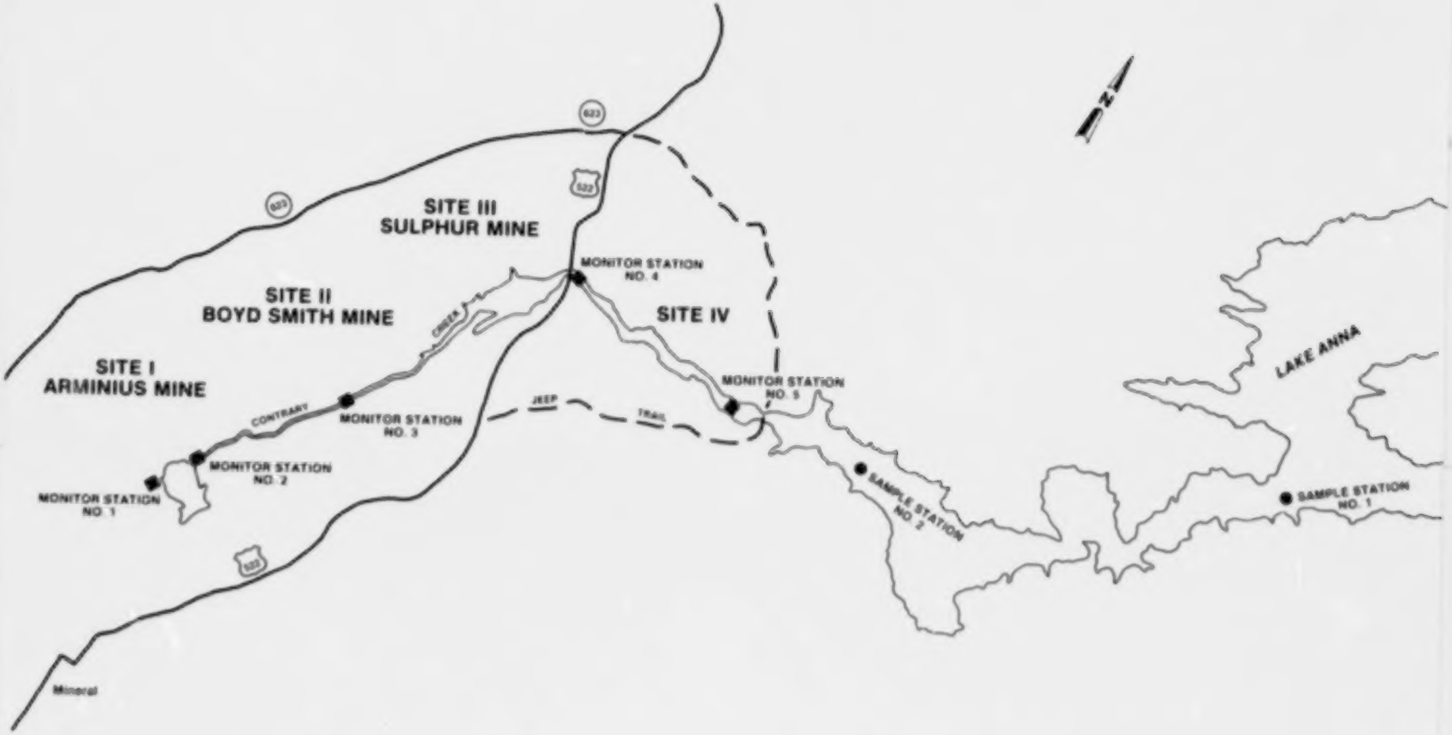
SITE III
SULPHUR MINE
SITE II
BOYD SMITH MINE
SITE I
ARMINIUS MINE
SITE IV
MONITOR STATION NO. 1
MONITOR STATION NO. 2
MONITOR STATION NO. 3
MONITOR STATION NO. 4
MONITOR STATION NO. 5
SAMPLE STATION NO. 1
SAMPLE STATION NO. 2
CONTRARY
CREEK
JEEP
TRAIL
LAKE ANNA
623
522
Mineral
N

Figure 2. Map of Contrary Creek Project site.

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a few seeps entered the creek above the water level, the majority of the seepage entered below the water level. Although usually these seeps were obscured by the creek water, they could sometimes be detected visually as dark brown pools clustering along the creek bank and in shallow pools.

Above water level seep pools were chemically analyzed and found to have pH's approaching 2.0. High metal concentrations were indicated by specific conductances in the range of 20,000 to 30,000 umhos/cm with individual metal analyses showing iron concentrations up to 16,200 ppm (Nordstrom and Dagenhart, 1979). These pools appeared to reach their maximum size during dry periods. The pools were flushed away during wet weather.

Nordstrom and Dagenhart found that during storm events peaks in the metal concentrations occurred (e.g. Fe, Cu, Zn, Al, etc.) with concurrent sharp decreases in pH. These peaks were attributed to the solubilization of water soluble sulfate minerals in the top layer of the mine spoil. During a heavy rainstorm, these minerals were mobilized by the surface runoff and made their way into the creek and into its smaller tributaries. The decrease in pH which occurred concurrently with the increase in metal concentrations resulted from the hydrolysis of iron in the ferric sulfate minerals (i.e. $Fe_2(SO_4)_3 + 6\ H_2O \longrightarrow 2\ Fe(OH)_3 + 3\ H_2SO_4$) and the flushing of accumulated acid from the first few centimeters of soil.

Nordstrom feels that the magnitude of the metal and acidity peaks and the concentrations of these variables are actually dependent on several factors. He believes that the length of the dry period preceeding a storm event directly influences the level of sulfate minerals formed in the upper spoil layers since longer dry periods allow for greater oxidation of sulfide minerals and subsequent formation of sulfates.

These sulfates may be deposited near the surface of the mine waste, after being carried upward in the interstitial water by capillary forces, and precipitate out upon subsequent evaporation of the interstitial water. Climatological conditions may also influence sulfate mineral deposition since this evaporation can be encouraged by high ambient temperatures, high wind speeds, and low relative humidities. Rainfall can also influence sulfate formation. A gentle, soaking rain may tend to saturate the mine waste and result in low levels of runoff to wash away the sulfates. This saturation, particularily when associated with high ambient temperatures, may increase the rate of sulfide metal oxidation. In lieu of rainfall, a steady high relative humidity can also increase the oxidation rate.

During storm events of significant intensity, decreases in the metal and acidity concentration peaks where noted at the Contrary Creek site (Nordstrom and Dagenhart, 1979). These decreases were attributed to the sulfate and acidic materials being almost completely flushed from the surface of the mine waste by surface runoff in combination with dilution of metal and acid concentrations by high volumes of water entering

Contrary Creek during storm events.

Reclamation of Contrary Creek

Reclamation of the Contrary Creek area began in the mid 1970's by clearing waste areas of brush, rubbish, etc. (Hinkle, 1980). This was followed by regrading the mining areas using bulldozers and endloaders. Diversion ditches and waterways were constructed concurrently with the regrading effort to reduce slopes and minimize water infiltration into the mine wastes. As much topsoil as possible was salvaged and spread over the wastes during this process.

Another portion of the reclamation effort consisted of excavating toxic materials from major portions of the creek's stream-bed and straightening the creek channel. Stone from a nearby granite quarry was then used as riprap to stabilize the banks of both Contrary Creek and portions of some of it's tributaries.

After regrading, lime and fertilizer were applied to the mine wastes (Hill et al., 1977). Lime was applied at a rate of nine tonnes per hectare (four tons per acre) and 10-10-10 fertilizer was applied at a rate of 1120 kg per hectare (1000 lbs per acre). The area was then overlayed with 10 cm (4 in) of municipal sewage sludge, obtained from the Blue Plains Sewage Treatment Plant in Washington, D.C., or with 38-0-0 fertilizer (ureaform) applied at a rate of 448 kg per hectare (400 lbs/acre). The areas receiving sludge were disked so that the sludge, lime and fertilizer were incorporated into the mine waste to a depth of approximately 8 to 16 cm (3-6 in). Rough areas were then dragged with heavy timbers to establish a smooth seed bed.

Initial seeding of the reclaimed areas took place during June and July of 1976 (Hinkle, 1980). Tall fescue (variation Kentucky Fescue-31) was applied at a rate of 67 kg per hectare (60 lbs per acre) while redtop and Ladino clover were applied at a rate of 5.6 kg per hectare (5 lbs per acre). Seeded areas were mulched with 5 tonnes per hectare (2 tons per acre) of small grain straw which was tied down in place by spraying with emulsified asphalt.

Due to a drought immediately after seeding and an abnormally cold fall, the seeding effort was not successful. Consequently, most of the Contrary Creek reclamation area was reseeded in the spring of 1977.

Prior to reseeding, a variety of fertilizers were applied to the site (Hinkle, 1980). Fertilizers used included 16-7-12, 18-18-6, 18-6-6, 10-10-10, and 38-0-0 and were applied at rates varying from 336.3 to 896.8 kg per hectare (300 to 800 lbs per acre). Reseeding then was performed using the grass species and application rates as follows: tall fescue applied at the rate of 67.3 kg per hectare (60 lbs per acre), redtop applied at the rate of 5.6 kg per hectare (5 lbs per acre), Ladino clover applied at the rate of 2.2 kg per hectare (2.0 lbs per acre), Korean Lespedeza applied at the rate of 11.2 kg per hectare (10 lbs per acre), and Weeping Lovegrass applied at the rate of 2.2 kg per hectare

(2.0 lbs per acre). In addition to the grass, 6500 Virginia Pines (Pinus virginiana) were planted at the Boyd Smith mine site while 17,000 Loblolly Pines (Pinus taeda) seedlings were planted at the Sulphur site.

For all practical purposes, this replanting effort was a failure due to the severe drought which occured from June to September of 1977. Only the drought-tollerant Weeping Lovegrass was able to be established in some areas by the fall of 1977. None of the other grasses planted survived and almost all of the pine seedlings died a few weeks after planting.

Consequently, in the fall of 1977, additional amendments (i.e. fertilizer, lime, and sludge) were applied as necessary and the area was reseeded using the seed types and application rates used in the spring of 1977, with the exception that Ladino clover was dropped from the mix. Reseeded areas were again mulched with small grain straw.

A combination of heavy rains and reapplication of sludge to the mine sites in November 1977, led to relatively successful results. However, the seeding effort failed in some areas particularly around the Sulfur Mine. From 1978 to 1980 maintenance soil amendment and seeding application were utilized as needed in troublesome areas. Irrigation was successfully used during dry periods after planting, and assisted in establishing vegetation. Tall fescue (Ky-31) was the most successful grass species. Weeping Lovegrass was also successful, particularly during dry periods due to its drought tolerance. Korean Lespedezia was moderately successful, while redtop and Ladino clover were found to be unsuitable for the Contrary Creek reclamation area.

By 1980, a viable cover soil (pH 6.5) was established in many areas due to the lime and sludge application. However, there were also some very toxic spots with pH's below those necessary for supporting vegetation. It has been postulated that extremely heavy applications of lime, fertilizer, and sludge will be necessary to establish plant growth in these areas (Hinkle, 1980).

Despite the relatively successful reclamation of the land in the Contrary Creek area, the water quality of Contrary Creek showed very little improvement (Hinkle, 1980). Before initiation of the reclamation effort, the quarterly pH averages ranged from 3.7 to 2.9. The quarterly pH averages in 1979 ranged from 3.6 to 3.0. Thus, no overall improvement in pH was noted. The same overall lack of success in the abatement of sulphate, copper, iron, lead, manganese, and zinc loads was also noted with 1979 levels being similar to those noted prior to reclamation.

Hinkle (1980) has postulated however, that if vegetation can be maintained, there should eventually be more surface uptake of moisture resulting in decreased water infiltration into the underlying pyrite wastes. This should subsequently result in less acid mine drainage leaching from stream banks into Contrary Creek and its tributaries.

This poor water quality was also reflected by the data obtained from biological studies of Contrary Creek which have shown the creek to be relatively devoid of aquatic life with little sign of improvement both prior to and after four years of reclamation. For the most part, only acid drainage tolerant species like iron bacteria, filamentous bacteria, certain green algae, blood worm midges, etc. have been noted in the creek (Hinkle, 1980). It is believed that in 1979 there may have been some slight diversification of the aquatic biological community in the upper portions of the creek. This diversification was likely the first step towards normal population of the creek and Hinkle (1980) consequently felt that it might have been the first sign of improvement in the aquatic life of the creek which resulted from the reclamation effort.

The cost of reclamation and maintenance at Contrary Creek through 1979 was \$14,170 per hectare (\$5,735 per acre). It was concluded (Hinkle, 1980) that sewage sludge should be used whenever feasible in the reclamation of areas adversely affected by mine wastes. Furthermore, he pointed out that the restoration of devastated mine sites could not be expected to be a short term project and that a significant period of time would be required to re-establish the natural environment.

UTILIZATION OF MUNICIPAL SLUDGE IN OTHER MINE RECLAMATION EFFORTS

The success in re-establishing vegetation on highly acid mine wastes using sewage sludge as a primary soil amendment has been shown to be beneficial in areas other than Contrary Creek. This method was used in restoring a portion of a 32.4 hectare (80 acre) plot in southeastern Tioga County, Pennsylvania (Miorin _et al_., 1979). This particular site included a portion of an inadequately restored coal strip mine overlying abandoned deep coal mine workings. Acid mine drainage resulted upon the exposure to air and water of the pyritic material which was intimately associated with the coal. The drainage discharged to a waterway, Morris Run, via three tributaries.

The site was regraded so that surface water was diverted from the deep mine workings to an adjacent stream. The site was then divided into two plots. On one 1.74 hectare (4.3 acre) plot, 1270 tonnes (1389 tons) of municipal sewage sludge (approximately 40% solids) were applied and spread to a depth of 8 cm (3.1 in) and then incorporated into the top 10 cm (4 in) of the mine waste. This resulted in an application rate of approximately 730 tonnes per hectare (326 tons per acre). No soil amendments other than the sludge were added to the plot.

The second plot was treated with ground agricultural limestone and fertilizer. The limestone was worked into the top 10 cm (4 in) of the plot. Both plots were then seeded with a mixture of tall fescue (Ky-31), birdsfoot trefoil, and common rye grass applied at rates of 39 kg per hectare (35 lbs per acre), 8 kg per hectare (7 lbs per acre), and 6 kg per hectare (5 lbs per acre), respectively. After seeding, the plots were mulched with old straw or hay at an application rate of 4.48 tonnes per hectare (4.9 tons per acre).

After approximately 1 year, vegetation was sampled from the sludge plot and the plot which received the ground limestone and fertilizer. The biomass of the grass (dried) from the sludge test plot was approximately three times greater than that obtained from the limestone/fertilizer plot. Miorin et al., (1979) attributed this difference to greater quantities of nutrients being available in the sludge plot.

Federal regulations relating to land application of sludge provide guidelines and limits on sludge application rates based on nutrient requirements of the vegetation as well as trace metal loadings. EPA has issued recommendations for the maximum amounts of trace metals that can be applied to agricultural soils via sewage sludge. These maximum amounts are related to the soil cation exchange capacity and are recorded in Table 1 (U.S. EPA, 1977).

TABLE 1. EPA RECOMMENDED MAXIMUM TRACE METAL LOADINGS FOR AGRICULTURAL LAND

Metal	Soil Cation Exchange Capacity (meg/100g)		
	0-5	5-15	15
	Amount of Metal (Kg/hc*)		
Pb	560.4	1120.8	2241.6
Zn	280.2	560.4	1120.8
Cu	140.1	280.2	560.4
Ni	140.1	280.2	560.4
Cd	5.6	11.2	22.4

*To convert Kg/hc to lb/ac multiply by 0.892179

Sludge has been shown to be excellent in increasing the organic humus of mined lands so that the nutrients in the humus are available for plant metabolism (Hinesley et al., 1972). The organic content of anaerobically digested sludge is close to that contained in the humus of naturally occurring soils. In addition, many trace elements necessary for plant growth are contained in sludges, however these same elements in high concentrations can be toxic to plant growth. Consequently, extremely high application rates of sludge to mined lands may result in increased soil phytotoxicity.

In addition to its other attributes, sludge application to mine wastes tends to increase the water holding and ion exchange capacities

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of the waste, thus creating a more favorable root zone for grasses and legumes (Grim and Hill, 1974).

It has been noted that the physical state of the sludge effects its performance in that partially dried sludge is often more effective in reclamation than is liquid sludge. At one reclamation site in West Virginia, both dried and liquid sludges were used (PEDCo Environmental, 1977). Plots treated with dried sludge yielded a grass cover in excess of 75% while revegetation efforts with plots treated with liquid sludge were unsuccessful. The liquid sludge apparently ran off sloped areas during periods of rainfall, thus its lack of success.

There was concern on the part of Miorin *et al.*, (1979) that the sludge used in the reclamation of the Tioga site might be a source of pathogenic microorganisms which could contaminate adjacent surface waterways. Consequently, at approximately 6 and 12 months after application of the sludge, microbial analyses were performed on water from an infiltration ditch surrounding the plot. Total coliform counts of 230 and 75 per 100 ml of water, respectively, were recorded. Due to these low counts, the sludge was judged not to represent a significant microbiological health risk. This assessment coincided with the evaluation of Dick *et al.*, (1971) who demonstrated the average die off of fecal coliforms in municipal sewage sludge to be 99.9% in 30 days.

Covering acidic mine spoil with non-acidic soil also has been used in reclamation efforts (PEDCo Environmental, 1977). This method has the obvious advantage of immediately providing a relatively high quality, mature soil base for plant growth. However, there is generally very little segregated native top soil in a mining area to be used in the regrading process since mine operators used to be extremely reluctant to segregate top soil from acidic spoil during mining (Grim and Hill, 1974).

If during the regrading process, top soil is placed as a final layer directly upon acidic mine spoil, the soil may become phytotoxic due to upward migration of acid and toxic metals. Grim and Hill (1974) advocated the placement of a neutral layer of fill between the spoil and the top soil to impede the migration of toxic factors into the root zone so that the establishment and subsequent growth of vegetation would not be inhibited. In addition, it has been recommended that on slopes to be covered with top soil the spoil be scarified to increase the bonding between the top soil and the spoil layer, and thus prevent slippage of the top soil (Dept. of Water Resources, State of Maryland, and Hittman Associates, 1972).

SECTION 4

MATERIALS AND METHODS

CLIMATOLOGICAL CONDITIONS

This study was conducted in the greenhouse of the U.S. EPA T&E pilot plant facility in Cincinnati, Ohio, from August 1980 through January, 1981.

During this six-month study period, climatological conditions in the greenhouse were adjusted to simulate, as nearly as possible, those conditions representative of the Contrary Creek study area during the months of May through October. Climatic controls were set based on climatological conditions that occurred in the study area from 1974 through 1978 as recorded by the National Oceanic and Atmospheric Administration (N.O.A.A.) in the form of Climatological Data Annual Summary Reports (N.O.A.A. 1974-1978).

The mean monthly precipitation that occurred in the study area from 1974 through 1978 is documented in Table 2. The precipitation pattern simulated during the six-month study period is indicated in this table as project simulated precipitation. Distilled, demineralized water was applied to the columns weekly according to statistics derived from the N.O.A.A. data (Table 3). Ten-year, 24-hour rain storm events were simulated during weeks 9, 18, 20, and 24 of the 24 week study period. Data derived from an isoplethic map illustrating rainfall occurring in the United States during 10-year, 24-hour events indicated that this occurrence in the Virginia study area resulted in approximately 15.2 cm (6 in) of rainfall (U.S. Dept. of Commerce, 1961).

The mean monthly temperature data were also calculated for the Contrary Creek study area using N.O.A.A. data for the above described five-year period and listed in Table 4 with the temperatures simulated during the study period so indicated. The temperature in the greenhouse was controlled using two heaters suspended from the overhead greenhouse superstructure. One heater was located in the southeast corner and the other in the northwest corner of the greenhouse (Figure 3). When necessary, cooling conditions in the greenhouse were achieved through ventilation using overhead roof vents along with side vents in the south and west walls of the greenhouse. In addition, the roof of the greenhouse was whitewashed to decrease solar radiation and to reduce the greenhouse temperature during the late summer months. The temperature control settings used in the greenhouse throughout the study period are also listed in Table 3. The greenhouse temperature was monitored using two wall thermometers located on the north and east walls of the greenhouse, in conjunction with a temperature probe attached to a continuously monitoring Rustrak Chart Recorder. Temperature readings from the wall thermometers were

TABLE 2: MEAN MONTHLY PRECIPITATION IN CENTIMETERS FOR THE STUDY AREA FROM 1974 THROUGH 1978

Month	1974	1975	1976	1977	1978	Five-Year Monthly Mean
January	6.32	8.43	9.27	4.34	21.7	10.0
February	3.96	5.94	3.91	0.97	0.74	3.05
March	8.38	16.4	7.21	6.22	10.3	9.70
April	6.05	4.62	4.09	4.65	9.32	5.75
May	8.79	8.53	8.18	3.63	12.1	8.25
June	11.1	26.7	11.6	3.86	14.6	13.6
July	13.7	20.2	6.81	5.21	13.7	11.9
August	6.96	7.80	10.9	5.33	21.0	10.4
September	13.9	24.0	10.7	5.18	5.99	12.0
October	0.58	4.83	22.3	41.5	2.92	8.43
November	4.60	5.13	3.66	14.7	6.55	6.93
December	13.1	9.86	4.80	12.6	9.22	9.92
Total yearly precipitation	97.4	142.	103	78.2	128	110

Project Simulated Precipitation

TABLE 3: CLIMATOLOGICAL CONDITIONS SIMULATED IN THE GREENHOUSE THROUGHOUT THE STUDY PERIOD

Month no.	Project week no.	Month simulated	Monthly temperature simulated	Monthly relative humidity	Weekly precipitation simulated (cm)
1	1	May	18°C	69%	2.06
	2				2.06
	3				2.06
	4				2.06
2	5	June	22°C	73%	3.40
	6				3.40
	7				3.40
	8				3.40
3	9	July	24°C	76%	18.2[a]
	10				2.98
	11				2.98
	12				5.18[b]
4	13	August	24°C	78%	2.60
	14				2.60
	15				2.60
	16				2.60
5	17	September	20°C	79%	3.02
	18				18.2[a]
	19				3.02
	20				18.2[a]
6	21	October	13°C	76%	2.11
	22				2.11
	23				2.11
	24				17.3[a]

[a] 10 yr, 24 hr rainstorm event simulated (15.2 cm of rainfall)
[b] 2.5cm of rainfall simulated

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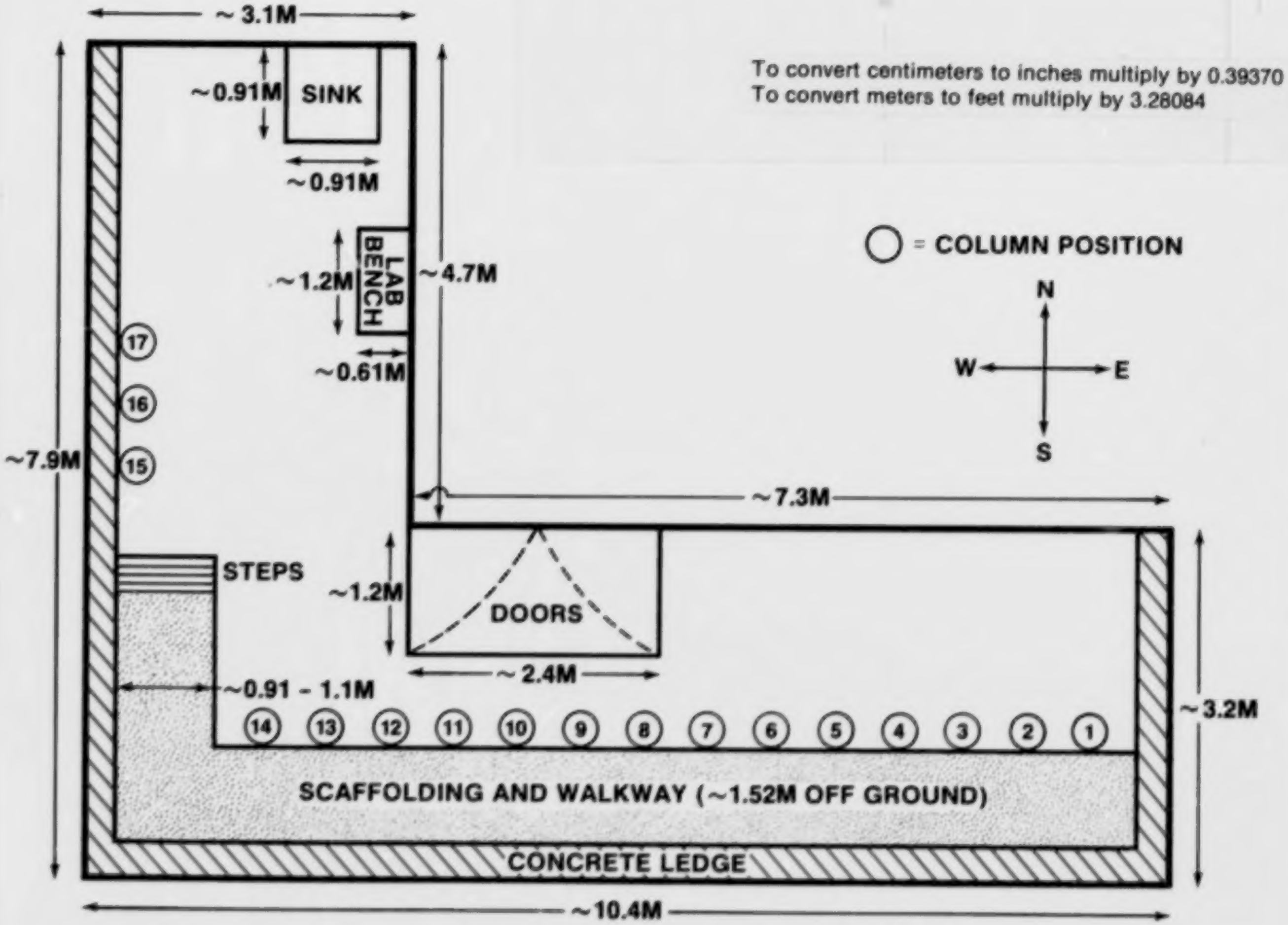
TABLE 4: MEAN MONTHLY TEMPERATURE IN °C FOR THE STUDY AREA FROM 1974 THROUGH 1978

Month	1974	1975	1976	1977	1978	Five-Year Monthly Mean
January	6.89	4.06	0.89	-3.89	-0.94	1.39
February	3.83	4.78	8.72	3.67	-1.50	3.89
March	10.1	6.78	11.2	11.4	6.39	9.17
April	14.7	11.2	14.6	14.9	13.2	13.7
May	18.1	19.0	17.2	19.2	17.1	18.1
June	20.1	22.2	22.4	21.3	22.2	21.7
July	23.4	23.4	23.7	25.9	23.2	23.9
August	23.2	24.3	22.6	24.9	24.4	23.9
September	19.4	18.7	19.6	21.4	21.1	20.1
October	13.6	15.9	11.8	13.0	13.3	13.5
November	9.06	11.2	5.39	9.50	9.56	8.94
December	4.56	3.22	1.94	2.56	5.06	3.44

Project Simulated Temperature


~ 3.1M
~0.91M
SINK
~0.91M
LAB BENCH
~1.2M
~4.7M
~0.61M
~7.9M
17
16
15
STEPS
~1.2M
DOORS
~2.4M
~0.91 - 1.1M
~7.3M
14
13
12
11
10
9
8
7
6
5
4
3
2
1
~3.2M
SCAFFOLDING AND WALKWAY (~1.52M OFF GROUND)
CONCRETE LEDGE
~10.4M
○ = COLUMN POSITION
N
W
E
S

To convert centimeters to inches multiply by 0.39370
To convert meters to feet multiply by 3.28084

Figure 3. Location of test columns in the greenhouse of the Test and Evaluation Facility.

recorded hourly and compared to the chart record.

Another climatological variable simulated during the study was relative humidity. Relative humidity is the actual water vapor of the air compared with the water vapor at saturation. The relative humidity control settings used during the study period were also based on N.O.A.A. data and are recorded in Table 3. Relative humidity in the greenhouse was regulated through the use of three Herrmidifier Model 500 Humidifiers. One humidifier was located in the upper northeast corner of the greenhouse facing the west wall, another was positioned in the far northeast corner facing the south wall with the third humidifier placed on the west edge of the north wall of the greenhouse. The relative humidity occurring in the greenhouse was monitored using a Sybron Wall Hygrometer located in the center of the north wall of the greenhouse. As with temperature, relative humidity measurements were read and recorded hourly.

Overall air circulation in the greenhouse was supplemented by two floor fans, one located against the east wall and another located against the far north wall.

ENGINEERING DESIGN

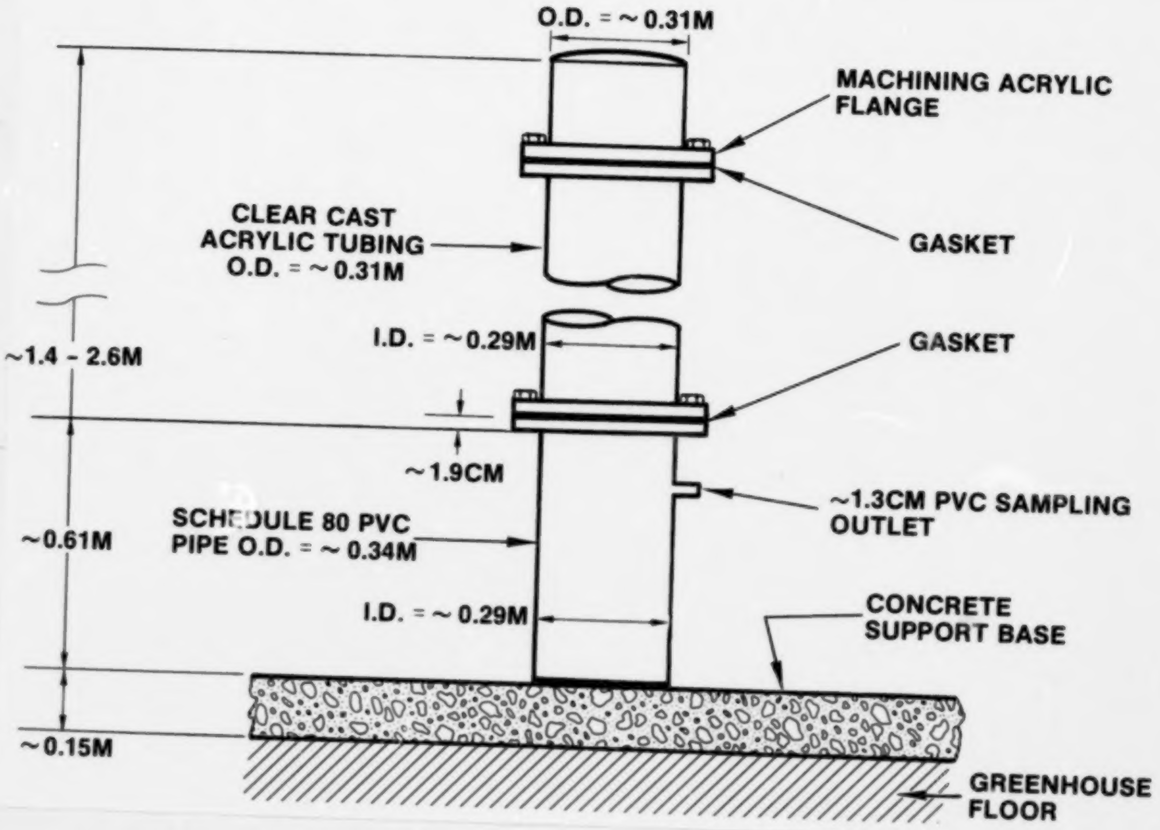
Column Design

The engineering design of this study included construction of seventeen 0.30-m (1-ft) diameter columns ranging in height from 2.1 m (7 ft) to 3.4 m (11 ft). Each column consisted of a 0.6-m (2-ft) section of 0.3-m (1-ft) diameter, schedule 80, polyvinyl chloride (PVC) pipe secured to a concrete support base. A lead support plate was placed between the PVC pipe and the concrete base. Attached to the upper end of the 0.6-m (2-ft) section of PVC pipe by means of a PVC/acrylic mounting flange was a piece of clear, cast acrylic tubing ranging in height from 1.4 m (4.5 ft) to 2.6 m (8.5 ft) (Figure 4). An additional machined acrylic flange was used to construct columns over 1.5 m (5 ft) in height.

Leachate Collection System Design

Located within this 0.6-m (2-ft) section of PVC pipe was an internal leachate collection system (Figure 5) consisting of a 0.3-m (12-in) length of 5.3-cm (2-in) diameter PVC pipe perforated with 0.32-cm (1/8-in) holes and wrapped with a 200-mesh stainless steel screen. The function of this screen was to prevent the discharge of fines from the ore or soil material into the leachate. A 5.1-cm (2-in) diameter, plain end PVC cap was used to seal the top of the system and through a series of PVC linkages (Figure 5) a 1.3-m (0.5-in) collection port allowed leachate to flow out of the column and into the external leachate collection system. This external system consisted of a test-tube air trap and an 18.9-liter (5-gallon) polyethylene collection vessel with a spigot for collection of leachate samples.

The internal leachate collection system was seated in concrete (Figure 5), the top layer of which was coated with an epoxy paint to prevent reaction


O.D. = ~0.31M
MACHINING ACRYLIC FLANGE
GASKET
CLEAR CAST ACRYLIC TUBING O.D. = ~0.31M
I.D. = ~0.29M
GASKET
~1.4 - 2.6M
~1.9CM
~1.3CM PVC SAMPLING OUTLET
SCHEDULE 80 PVC PIPE O.D. = ~0.34M
~0.61M
I.D. = ~0.29M
CONCRETE SUPPORT BASE
~0.15M
GREENHOUSE FLOOR

To convert centimeters to inches multiply by 0.39370
To convert meters to feet multiply by 3.28084

Figure 4. Column design.

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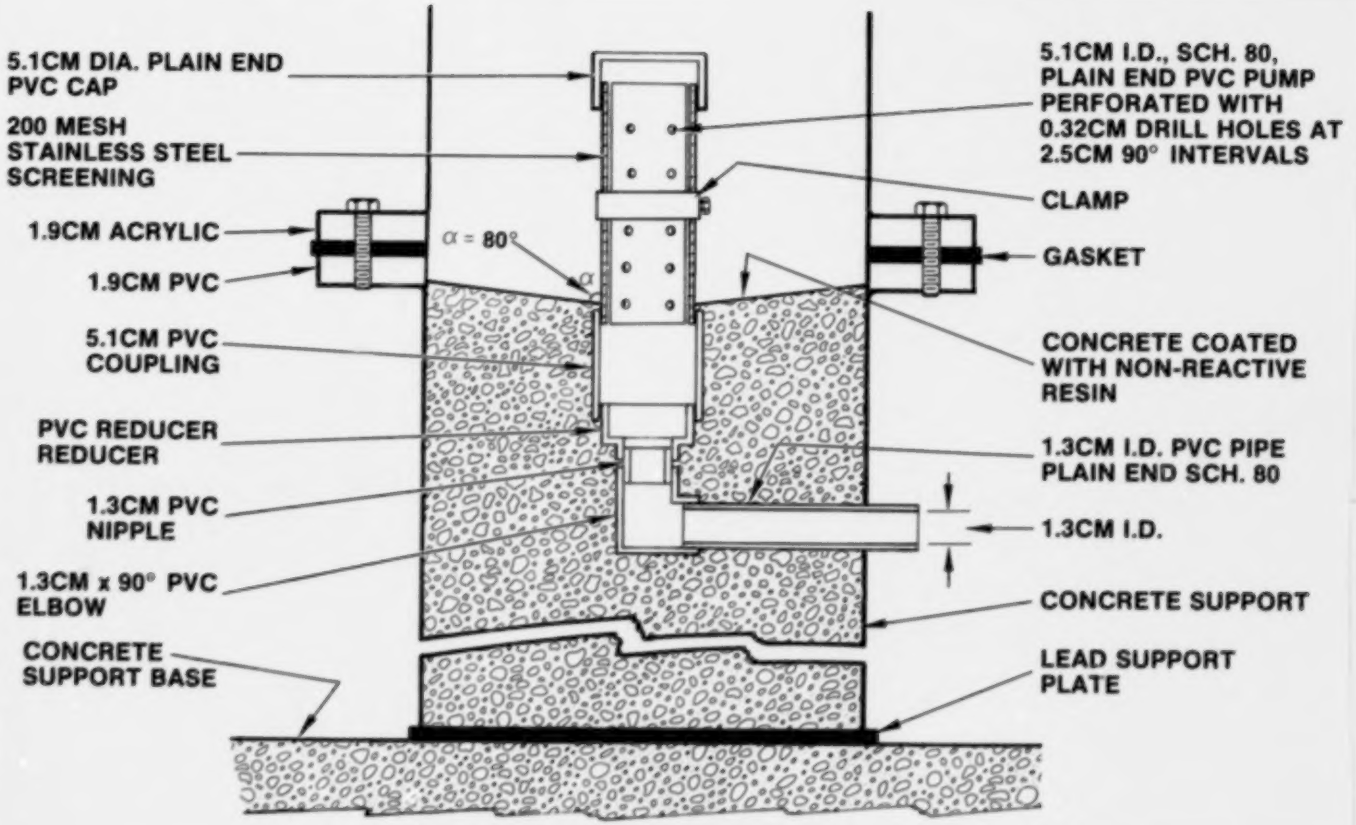
5.1CM DIA. PLAIN END PVC CAP
200 MESH STAINLESS STEEL SCREENING
1.9CM ACRYLIC
1.9CM PVC
5.1CM PVC COUPLING
PVC REDUCER REDUCER
1.3CM PVC NIPPLE
1.3CM x 90° PVC ELBOW
CONCRETE SUPPORT BASE
α = 80°
α
5.1CM I.D., SCH. 80, PLAIN END PVC PUMP PERFORATED WITH 0.32CM DRILL HOLES AT 2.5CM 90° INTERVALS
CLAMP
GASKET
CONCRETE COATED WITH NON-REACTIVE RESIN
1.3CM I.D. PVC PIPE PLAIN END SCH. 80
1.3CM I.D.
CONCRETE SUPPORT
LEAD SUPPORT PLATE

To convert centimeters to inches multiply by 0.39370
To convert meters to feet multiply by 3.28084

Figure 5. Internal leachate collection system.

with the acidic leachate. At the base of the 5.1-cm (2-in) diameter PVC pipe, the concrete was sloped at an 80° angle relative to the pipe to enhance leachate collection.

The experimental design of this project involved the construction of a total of 17 columns in the greenhouse (Figure 6). Fourteen of the columns were constructed opposite the north wall of the greenhouse to allow a southerly exposure, while the remaining three columns received a western exposure. Access to the tops of the columns for observation and monitoring of vegetative growth was made possible by construction of a 1.2-m (3.5-ft) wide steel walkway approximately 1.5 m (5 ft) above the floor of the greenhouse. By placing the columns on the north side of the walkway, the scaffolding also provided support in addition to access.

EXPERIMENTAL DESIGN

Column Loading

Approximately 3.2 tonnes (3.5 tons) of the pyritic mine solid waste were obtained from the Contrary Creek site at a location on the east bank of the creek, upstream from the dam (Figure 2). In addition, approximately 1.8 tonnes (2 tons) of cover soil (topsoil) native to the Contrary Creek site were transported to the T&E Facility. In preparation for screening, the materials were spread out separately and air-dried for a four to five day period. Bulk screening of each material was done with a 1.3-cm (0.5-in) square screen. Less than 0.1% of the mine waste collected was found to consist of material greater than 1.3 cm in size. This larger material was broken manually so as to pass through the screen.

Soil tests were performed on representative samples of these materials according to the procedures outlined by Sobek (Sobek et al., 1978). Specific tests performed are shown in Table 5. In addition, the particle size distribution of a representative sample of the mine waste collected was then determined by a standard sieve analysis (Lambe, 1951). Chemical analyses were also performed on the mine waste and cover soil materials and included a combination acid extraction of each material. The extraction procedure used was a modification of the ASTM (American Society for Testing and Materials) standard method D 2013 (ASTM, 1978) and involved placing 0.2 g of the material into a beaker containing 20 ml of freshly prepared aqua regia and 20 ml of concentrated hydrofluoric acid. Each sample was heated to dryness at 160°C, reconstituted using concentrated nitric acid and distilled water, and analyzed by Inductively Coupled Argon Plasma emission spectroscopy (ICAP). Concentrations of iron, lead, copper and zinc were quantified and recorded.

After being air dried for approximately four days the materials were passed through a 1.3-cm (0.5-in) screen. The mining solid waste was then uniformily mixed, as nearly as possible, and placed in the columns in 0.30-m (12-in) lifts. The mine waste was then graded, and mechanically vibrated until the material was compacted to approximately 1600 kg/m^3 (100 lb/ft^3). The cover soil was placed in the columns in the same manner and compacted to a density of approximately 800 kg/m^3 (50 lb/ft^3). The mass of mine waste and

TABLE 5: MINING SOLID WASTE AND COVER SOIL ANALYSIS

Variable Analyzed	Mining Solid Waste	Cover Soil
Slurry pH	1.8	5.1
Double Acid Extractable Potassium (ppm by wt.)	3.69	59.8
Double Acid Extractable Calcium (ppm by wt.)	146	575
Double Acid Extractable Magnesium (ppm by wt.)	174	67.9
Double Acid Extractable Phosphorus (ppm by wt.)	73.2	3.64
Double Acid Extractable Sodium (ppm by wt.)	76.8	0
Organic Carbon (Walkley - Black Method)	2.2%	4.3%
Lime Requirement (S.M.P. Buffer Method)	24 tonnes limestone/ 1100 tonnes material[1]	2.2 tonnes limestone/ 1100 tonnes material[1]

[1] one tonne (metric) is equal to 1000 kg or to approximately 2200 pound mass

TABLE 6. COLUMN CONTENTS

COLUMN NUMBER	PYRITE MINING SOLID WASTE (KG)	COVER SOIL MATERIAL (KG)	AGRICULTURAL LIMESTONE ADDED AS AN AMENDMENT (KG)	AGRICULTURAL LIMESTONE USED AS A LAYER (KG)	MUNICIPAL SLUDGE (DRY WT.) (KG)	15-15-15 COMERCIAL FERTILIZER (G)	TALL FESCUE KY-31 (G)	WEEPING LOVEGRASS (G)
1	153	0	3.4	0.0	0.0	8.17	0.49	0.016
2	147	0	3.2	0.0	0.0	8.17	0.49	0.016
3	0	90	0.18	0.0	0.0	8.17	0.49	0.016
4	0	92	0.18	0.0	0.0	8.17	0.49	0.016
5	156	0	3.4	0.0	5.15	8.17	0.49	0.016
6	138	0	3.0	0.0	5.15	8.17	0.49	0.016
7	140	0	3.1	0.0	5.15	8.17	0.49	0.016
8	145	0	3.2	0.0	5.15	8.17	0.49	0.016
9	128	34	0.07	0.0	0.0	8.17	0.49	0.016
10	135	13	0.03	0.0	0.0	8.17	0.49	0.016
11	148	44	0.09	0.0	0.0	8.17	0.49	0.016
12	141	46	0.10	0.0	0.0	8.17	0.49	0.016
13	145	90	0.18	0.0	0.0	8.17	0.49	0.016
14	132	91	0.18	0.0	0.0	8.17	0.49	0.016
15	137	46	0.09	1.6*	0.0	8.17	0.49	0.016
16	147	50	0.10	0.0	0.0	8.17	0.49	0.016
17	146	46	0.09	3.2**	0.0	8.17	0.49	0.016

*REPRESENTS 1 IN. LAYER APPROXIMATELY

**REPRESENTS 2 IN. LAYER APPROXIMATELY

cover soil placed in each column were recorded (Table 6).

Amendments were then added to the columns. The types and application rates of amendments used in this column study were similar to those used at the full-scale demonstration site at Contrary Creek and included agricultural limestone, 15-15-15 commercial chemical fertilizer, and anaerobically digested sewage sludge obtained from the Cincinnati Municipal Wastewater Treatment Facility. The organic carbon content and predominant metal composition of this sludge are listed in Table 7. The type of agricultural limestone used in this study was Plum Run Superfine Agricultural Limestone from the Plum Run Plant in Peebles, Ohio. According to the manufacturer, the minimum guaranteed chemical analysis of the limestone included 19% calcium and 11% magnesium. The total neutralizing power in terms of calcium carbonate was 95%.

TABLE 7: METAL AND ORGANIC CARBON ANALYSIS OF SLUDGE

Variable analyzed	Concentration (mg/g dry wt)
Copper	9.47
Nickel	1.35
Iron	122
Zinc	7.16
Cadmium	0.22
Lead	8.12
Moisture content	78%
Organic carbon	56.3%

[a] Metals were analyzed by atomic absorption spectroscopy organic carbon analysis by low temperature ignition

Surface Treatment Techniques Studied

Overall, four surface treatment techniques for disposal of this pyrite mine solid waste were examined throughout this study. These techniques were: (1) the incorporation of sewage sludge, agricultural limestone and 15-15-15 commercial fertilizer into the upper strata of the mine waste, (2) the layering of sewage sludge and fertilizer over the lime-treated waste, (3) the placement of a cover soil layer over the mine waste, and (4) the placement of a limestone layer positioned between the mine waste and cover soil layer.

Columns 1 and 2 contained 1.2 m (4 ft) of the pyrite mine waste and served as controls. Columns 3 and 4 contained 1.2 m (4 ft) of the cover soil material and also served as controls.

Columns 5 and 6 simulated the reclamation work done at the Contrary Creek Demonstration site. These columns contained the sludge, lime and fertilizer incorporated into the top 0.15 m (0.5 ft) of the 1.2-m (4-ft) column of pyrite mine waste. Columns 7 and 8 contained these same materials in a similiar configuration. A 1.2-m (4-ft) layer of the mine waste was placed in each column. Into the top 0.15 m (0.5 ft) of the mine waste was incorporated lime only. The fertilizer and sludge were mixed and layered on top of the lime-treated mine waste.

Columns 9 through 14 addressed the effect of a layer of cover soil placed over the mine waste. Columns 9 and 10 contained 1.2 m (4 ft) of the mine waste under 0.46 m (1.5 ft) of cover soil. Similarly, columns 11 and 12 contained a 0.61-m (2-ft) layer of cover soil placed on top of 1.2 m (4 ft) of the mine waste while columns 13 and 14 contained 1.2 m (4 ft) of mine waste under 1.2 m (4 ft) of cover soil. Columns 15 through 17 were used to examine the effects of placing a neutralizing layer between the cover soil and mine waste. Column 15 contained 0.61 m (2 ft) of the mine waste, on top of which was placed 1.6 kg (3.5 lb) of lime (approximately 2.5 cm (1 in) in height). Positioned over the lime was a 0.61-m (2-ft) layer of cover soil. Column 17 was loaded in a similar manner and contained 3.2 kg (7 lb) of lime (approximately 5.0 cm (2 in)) layered between a 0.61-m (2-ft) layer of cover soil and 0.61-m (2-ft) layer of mine waste. Column 16 contained 0.61 m (2 ft) of mine waste under 0.61 m (2 ft) of cover soil and served as a control. The contents of each column are shown schematically in Figure 6 and listed in Table 6. It can be noted from Table 6 that 15-15-15 commercial chemical fertilizer was applied equally to the surface of all columns at an application rate of approximately 1.1 tonnes/ha (0.5 tons/ac) and municipal sewage sludge at an application rate of approximately 709 tonnes/ha (315 tons/ac). The application rate of agricultural limestone was based on laboratory soil tests performed on representative samples of the mine waste and cover soil (Table 4) and averaged approximately 440 tonnes/ha (196 tons/ac) for the pyrite mine waste, and approximately 17.1 tonnes/ha (7.6 tons/ac) for the cover soil material.

Column Preparation

The contents of all 17 columns were initially saturated to aid in the establishment of more uniform conditions among the columns. A two-part saturation procedure was used to prevent acidity and metals present in the mine waste from artifically contaminating the cover soil material.

The mine waste in the columns was wetted initially by introducing distilled water through the bottom of each column containing the mine waste (all columns except 3 and 4). The water was contained in 19 liter (5 gallon) holding tanks located adjacent to the bottom of the columns. A low stream of water was pumped through Tygon tubing into the internal leachate collection system (Figure 5) and into the center of the column of mine waste. The water

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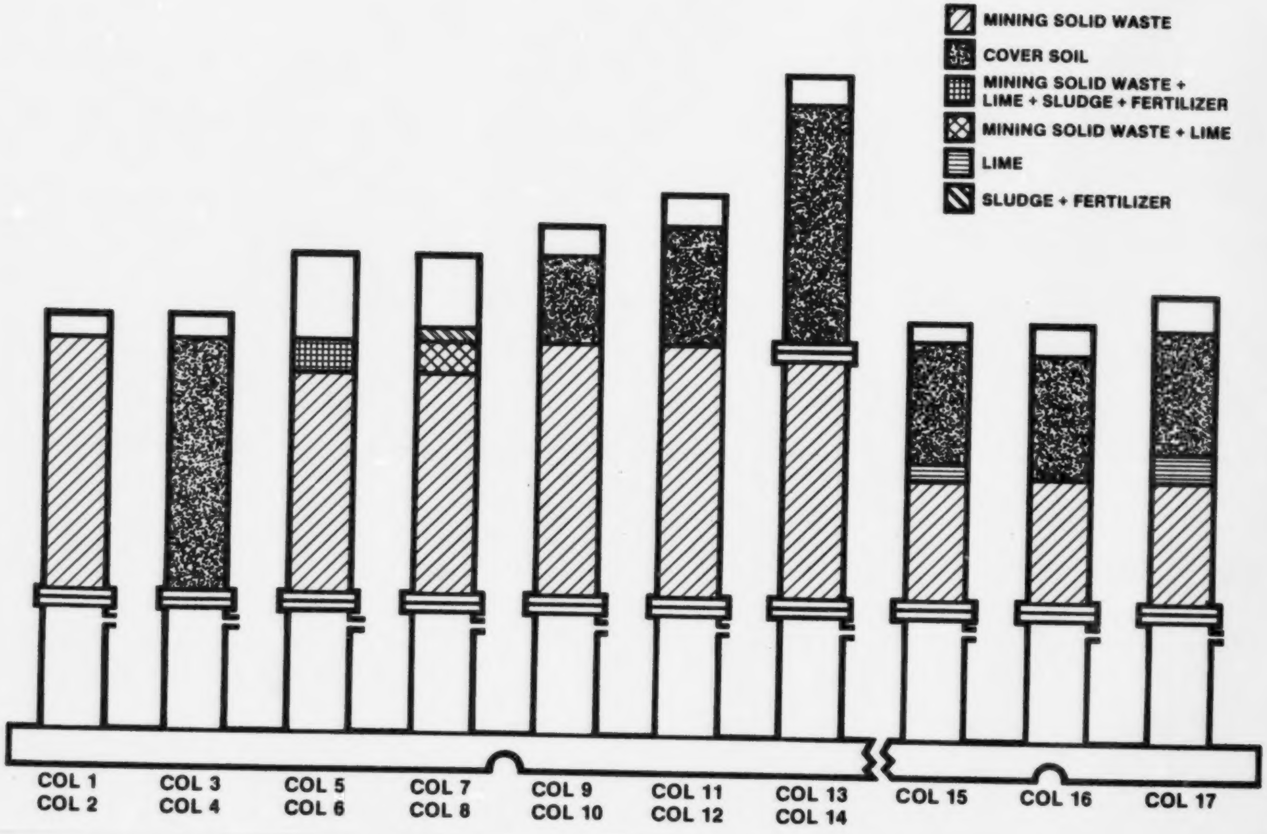
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COVER SOIL
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⊠ MINING SOLID WASTE + LIME
▤ LIME
▧ SLUDGE + FERTILIZER
COL 1
COL 2
COL 3
COL 4
COL 5
COL 6
COL 7
COL 8
COL 9
COL 10
COL 11
COL 12
COL 13
COL 14
COL 15
COL 16
COL 17

Figure 6. Contents of columns 1 through 17.

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was pumped at low flow rates and allowed to soak through the mine waste until a 10-cm (4-in) head of water was obtained above the mine waste (in columns 1, 2, and 5 through 8) or until the water had visibly reached the mine waste/cover soil interface (in columns 9 through 17). At this time the bottom discharge valves on the columns were closed.

The cover soil in the columns was then wetted by introducing distilled water to the top of the columns. The water was contained in the same holding tanks used previously and allowed to soak through the cover soil layers by means of gravity flow. Saturation was determined in the same manner used for the mine waste.

The columns were maintained under saturated conditions for approximately 24 hours, after which time the bottom valve on each column was opened and the columns were allowed to drain by gravity flow for an additional 24 hours. The quantity of water needed to saturate each column and the quantity of leachate collected were recorded.

Finally, the columns were seeded with a mixture of 'Kentucky-31' Tall Fescue (seeding rate 67.3 kg/ha (60 lb/ac)) and Weeping Lovegrass (seeding rate 2.2 kg/ha (2 lb/ac)). The Weeping Lovegrass (*Eragrostis curvola*) seeds were obtained from The Wetsel Seed Co., Inc. and had a purity of 97.95% and germination of 85.00%. The Kentucky Fescue (*Festuca arundinacea*) seeds were obtained locally and characterized by a purity of 75% and germination of 85%. These grasses also had been successfully used at these application rates at the Contrary Creek site. Finally, the columns were wrapped with several layers of brown paper to prevent light penetration into the subsoil.

METHODS USED FOR DATA ANALYSIS AND CHARACTERIZATION

After the columns had been loaded and saturated, simulation of the climatological conditions of the study area, previously discussed, began in the greenhouse (Table 3). Distilled water was added to the columns weekly to simulate various rainfall conditions. Approximately 24 hours after the water was added to the columns, leachate samples were collected from each column and analyzed for pertinent physical and chemical properties.

Acidity, pH, specific conductance, iron, copper, lead, and zinc in the leachate were quantified using EPA methods (U.S. EPA, 1979). The resulting data were then converted to load data (i.e., concentration times flow) and these data were subjected to statistical analyses.

Since all of the test columns were duplicated (Figure 4), a paired observation t-test was used to determine whether the means of the data from the columns could be considered from the same population. The calculated t values were found in all cases to be less than the tabular value at the 95% confidence level; therefore, the data from the duplicates were combined and considered as one. The same statistical procedure was repeated to compare the control columns to the experimental columns. In each case, the calculated t statistic exceeded the tabular value at the 95% confidence level; therefore, it was concluded that the characteristics of the experimental columns were indeed different from those of the control columns. These calculations were performed using a digital computer utilizing the SPSS (Statistical Package for the Social Sciences) (Nie et al., 1975) statistical package.

Vegetative growth was measured periodically during the project. At the end of the study period, samples of the vegetation were and analyzed for metal uptake. Total metal concentrations of copper, iron, lead, and zinc were determined by Atomic Absorption Spectrophotometry after a nitric acid digestion was performed. Samples were cut into small pieces placed into beakers and dried in an 80°C oven overnight. Approximately 0.5 gram of each sample was digested with redistilled nitric acid and brought to a 25 ml end volume. Metal concentrations were determined from these 25-ml samples and recorded.

In a similar manner, mine waste and cover soil samples were also taken and analyzed at the end of the study period to assess metal migration through the mine waste and cover soil material. As each column was disassembled, a sample from each approximately 0.31 m (1 ft) of material was obtained. Total concentrations of the above mentioned metals were determined in accordance with procedures outlined by Sobek (Sobek et al., 1978) and recorded (Appendix E).

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SECTION 5

RESULTS

MINE WASTE AND COVER SOIL ANALYSIS

As received at the T&E Facility, the pyritic mine waste was appreciably moist with a water content of approximately 11 percent. The moisture content of the cover soil was also measured using standard soil procedures (Lambe, 1951) and determined to be approximately 22 percent.

After bulk screening each material with a 1.3 cm (0.5-in) square screen, the particle size distribution of the mine waste was determined by a sieve analysis (Lambe, 1951) with results illustrated in Figure 7. In accordance with the M.I.T. grain size classification system, soil diameters between 0.002 and 0.6 mm (0.00008 and 0.02 in) were classified as silt, and diameters between 0.06 and 2.0 mm (0.002 and 0.8 in) were classified as sand. Based on this system the particle size distribution of the mine waste was approximately 1-percent gravel, 24-percent sand, 73-percent silt, and 2-percent clay.

Additional soil tests were performed on the mine waste and cover soil material to determine the capability of these materials to support vegetative growth. These tests included soil pH, double acid extractable nutrients (potassium, calcium, magnesium, phosphorus and sodium), total organic carbon and lime requirement. It was determined the mine waste had a pH of approximately 1.8 and was extremely acidic compared to the cover soil material which had a pH of approximately 5.1. This was further exemplified in the lime requirement of the mine waste which was approximately eleven times that of the cover soil material. Overall, the mine waste was low in potassium and calcium, and high in magnesium, phosphorus and sodium compared to the cover soil material (Table 5). The organic carbon content of the mine waste was approximately one-half that of the cover soil material. Further chemical studies were performed with the mine waste and cover soil material in order to determine initial concentrations of variables considered to be important in the column studies. The samples selected for this work consisted of representative samples of each material obtained after initial bulk screening. After being digested with aqua regia and hydrofluoric acid, the samples were analyzed by ICAP. The data obtained showed the pyrite mine waste contained approximately 200 mg/g iron, 4 mg/g lead, 10 mg/g copper, and 2 mg/g zinc. This compared to approximately 40 mg/g iron, 1 mg/g copper, less than 1 mg/g lead, and less than 1 mg/g zinc present in the cover soil material.

ANALYSIS AND CHARACTERIZATION OF THE LEACHATE

Leachate samples from each column were collected weekly and analyzed for

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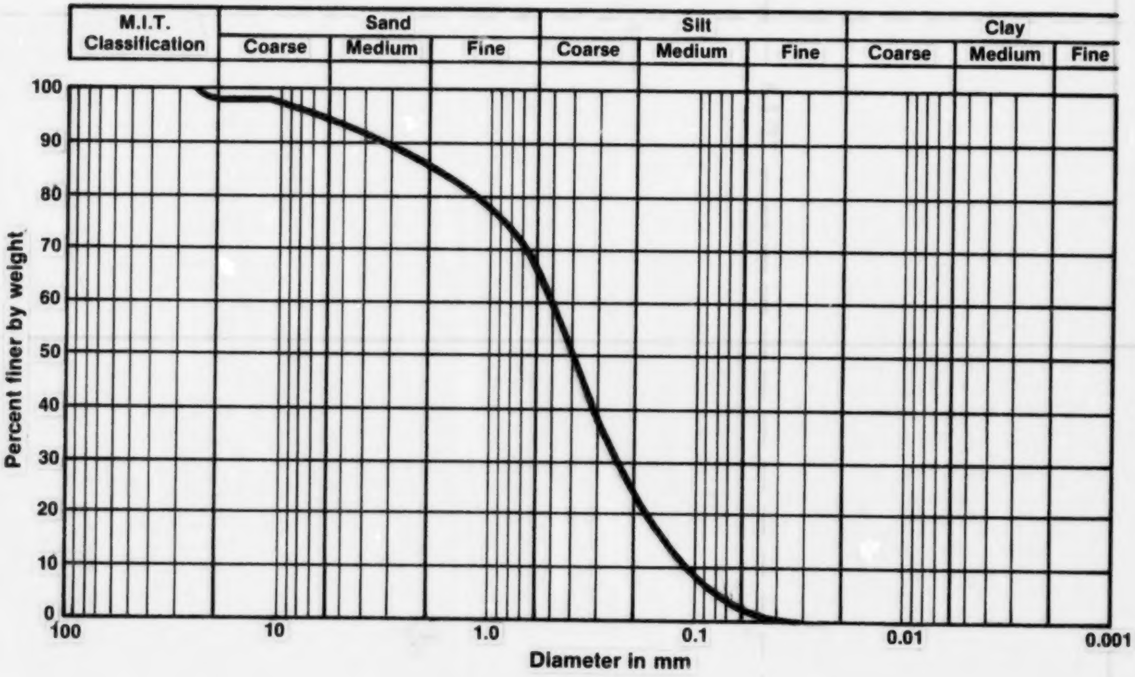

Figure 7. Particle size distribution of the mine waste.

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pertinent physical and chemical properties. Quantities of leachate collected and leachate flow rates were measured and recorded (see Appendix A).

Immediately after being collected, the temperature, pH, turbidity, specific conductance, and acidity of the samples were quantified and recorded (see Appendix B) using previously described methods. In addition, iron, copper, zinc and lead concentrations in the leachate were determined for each column (see Appendix C). The metal and acidity data were converted to load data (concentration times flow) and these data were then subjected to the statistical and computer analyses described earlier. The characteristics of the experimental columns were then compared to the control columns and the results of each treatment technique evaluated.

COLUMN LEACHATE pH AND ACIDITY DATA

pH Data

Figures 8 through 24 illustrate leachate pH as a function of time for each of the columns. Overall, the pH of the leachate generated from the mine waste control columns (columns 1 and 2) ranged from 1.2 to 2.1 (Figures 8 and 9), while leachate from the cover soil control columns (columns 3 and 4) had a near neutral pH throughout the study period (Figures 10 and 11).

The pH of the leachate from the experimental columns containing lime, sludge and fertilizer incorporated into the upper strata of the mine waste (columns 5 and 6) was approximately 1.4 initially, and increased steadily throughout the study period (Figures 12 and 13) with a final pH of approximately 3.3 being measured. In comparison, the pH of the leachate generated from the columns containing the same materials layered (columns 7 and 8) was also 1.4 initially, but measured 1.9 at the end of the study period and was never greater than 2.5 throughout the study period (Figures 14 and 15). In addition, the rate of increase in the pH over time, as indicated by the slope of these graphs, was almost four times greater for the columns containing the materials incorporated compared to those columns containing the materials layered.

When a layer of cover soil was placed over the mine waste, a dramatic change in the pH of the resulting leachate was realized. The most significant change in leachate pH was generated from columns containing 1.2 m (4 ft) of cover soil material placed over the mine waste (columns 13 and 14). The pH of leachate from these columns steadily increased from an initial pH of 1.4 to a final pH of approximately 3.0 at the end of the study period (Figures 20 and 21). In comparision, the pH of leachate from the columns containing 0.61 m (2 ft) of cover soil (columns 11 and 12) ranged from approximately 1.4 initially to 3.2 recorded during week 18 of the study period. Even though the maximum pH attained in these columns was higher than that attained in the columns containing twice as much cover soil (columns 13 and 14), the rates of change in pH over the study period were different for the two column sets. The rate of change in leachate pH of the columns containing 0.61 m (2 ft) of cover soil (columns 11 and 12) was less than one-third that of the columns containing 1.2 m (4 ft) of cover soil. This was

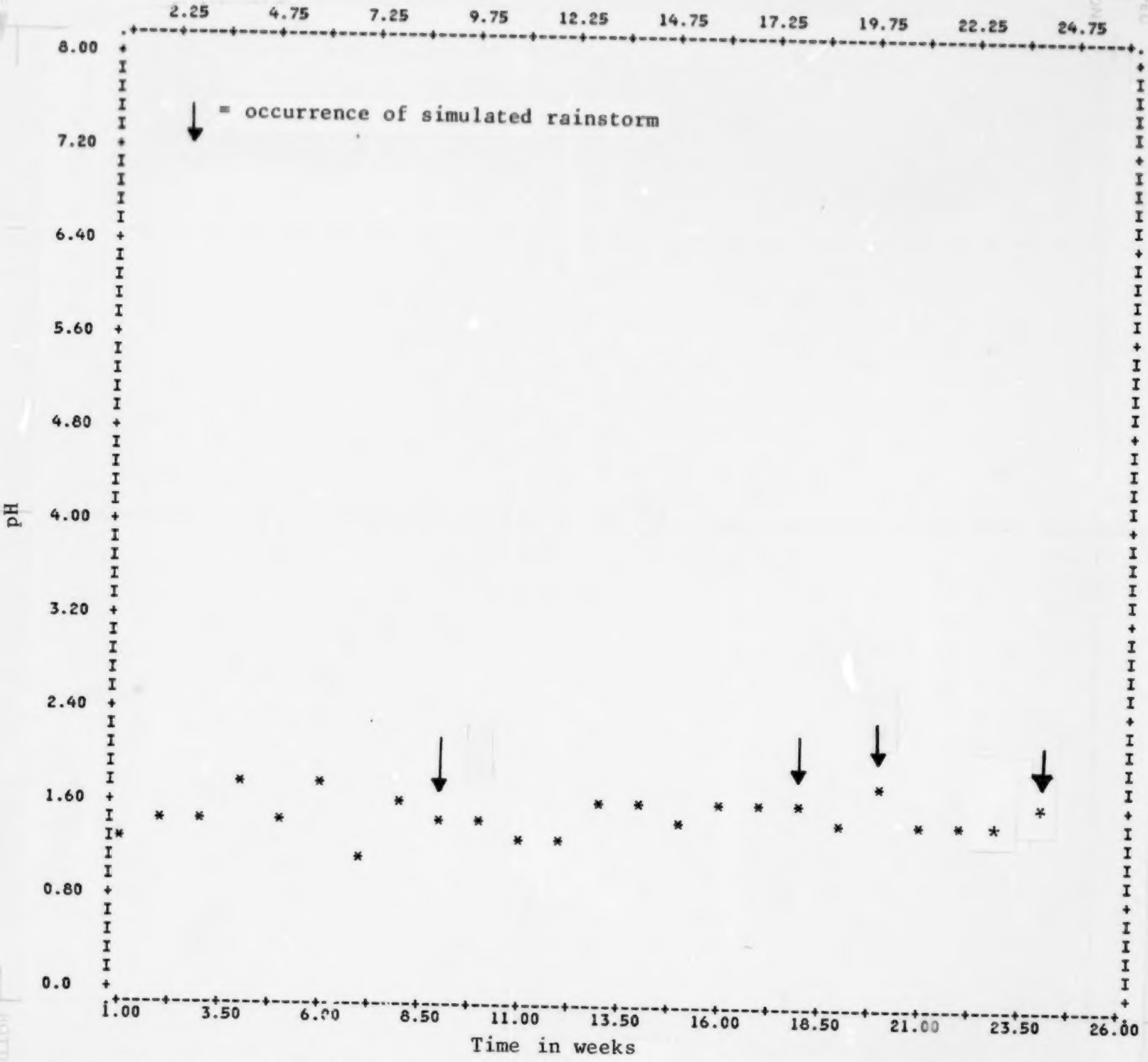

Figure 8. Graph of pH as a function of time for Column 1 (mine waste control).

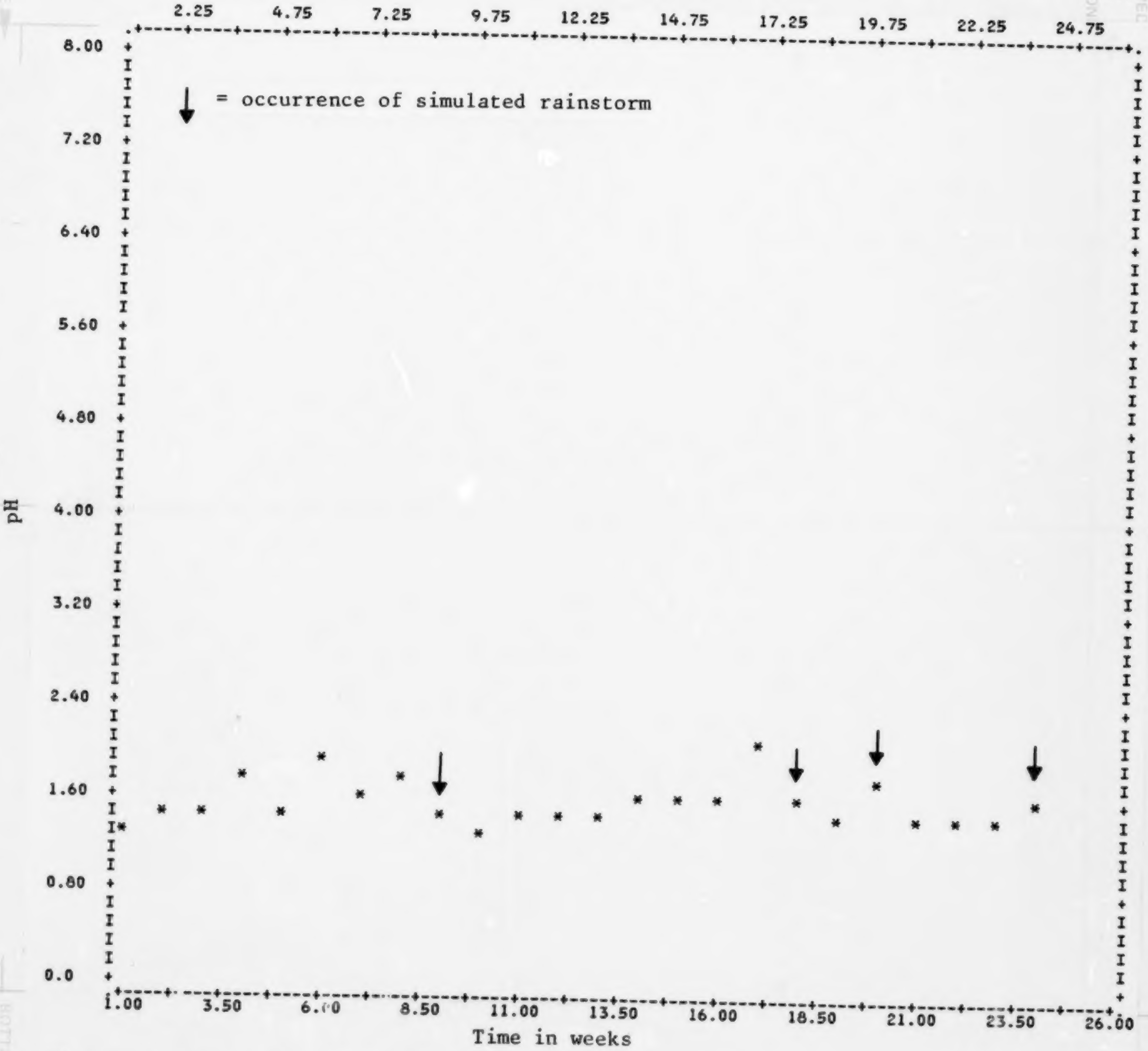

Figure 9. Graph of pH as a function of time for Column 2 (mine waste control).

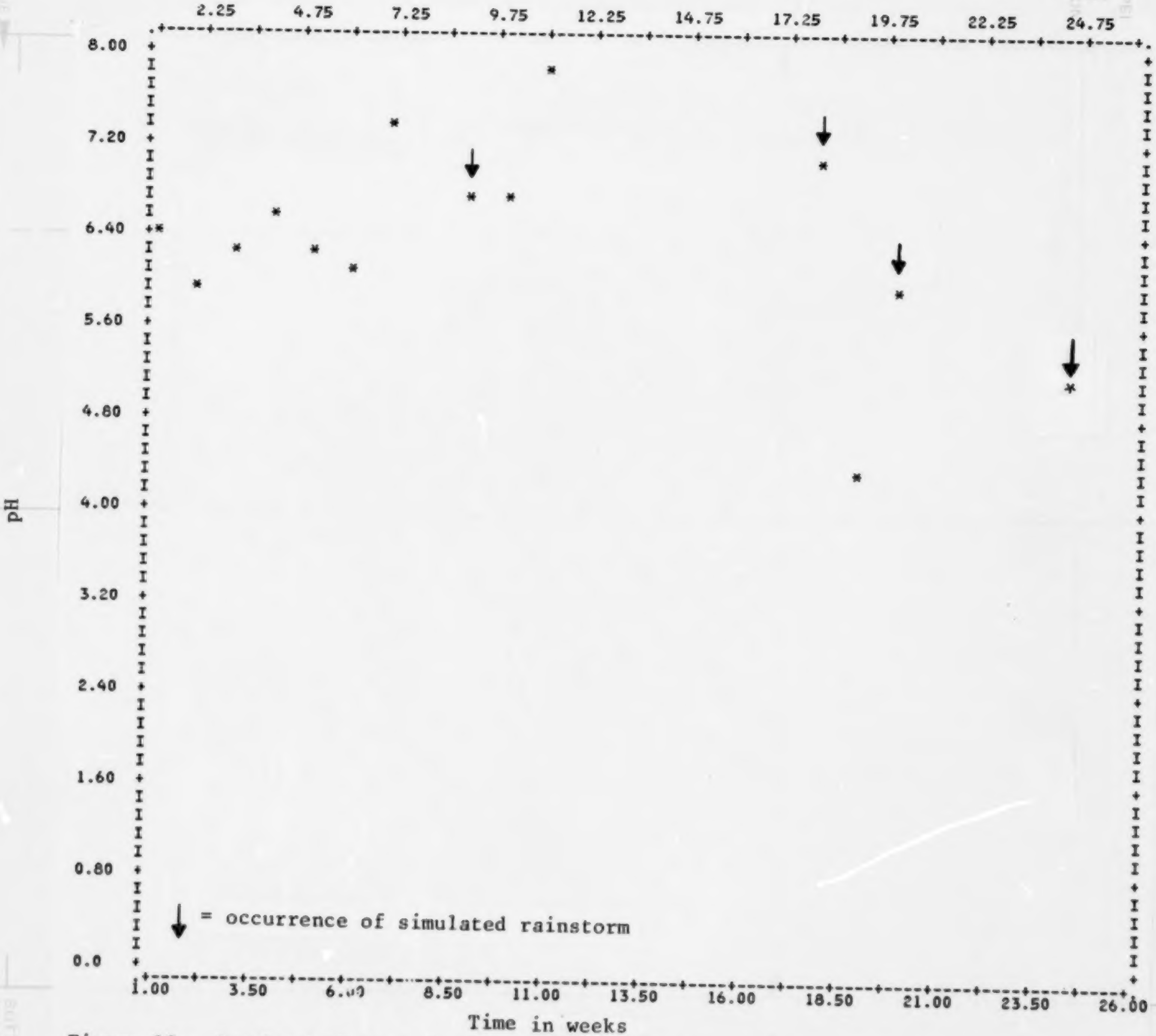

Figure 10. Graph of pH as a function of time for Column 3 (cover soil control).

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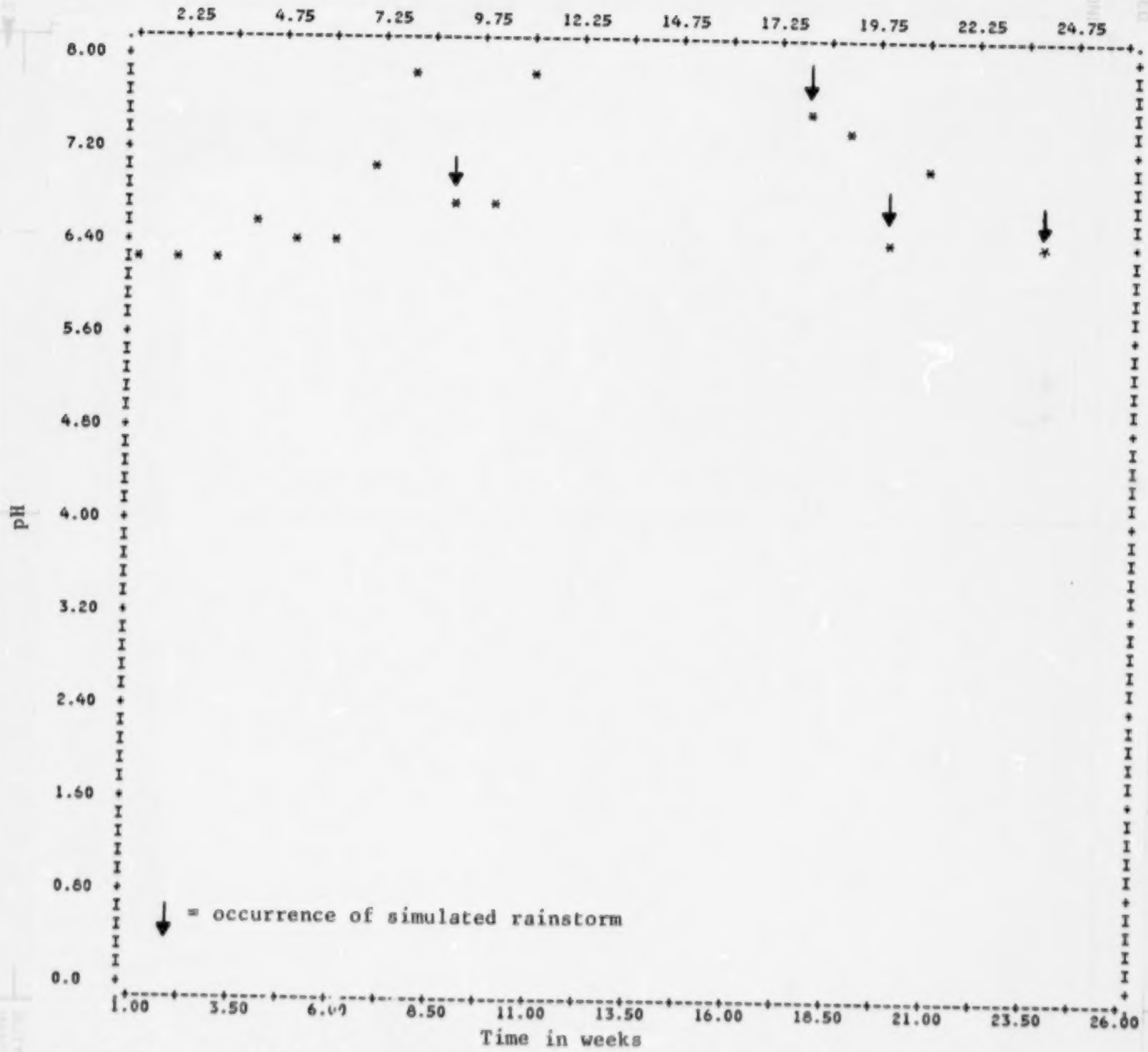

Figure 11. Graph of pH as a function of time for Column 4 (cover soil control).

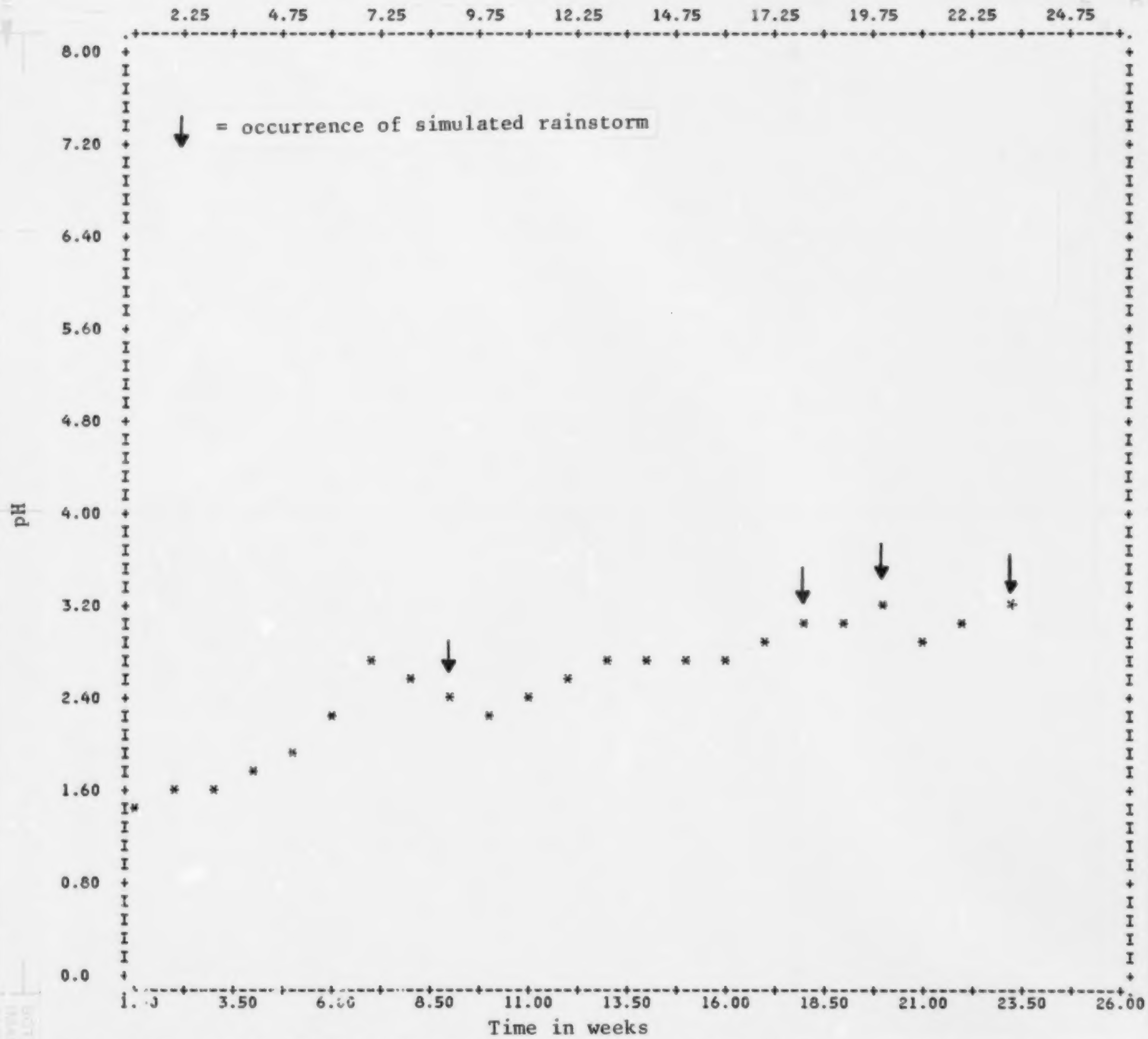

Figure 12. Graph of pH as a function of time for Column 5 (incorporated amendments).

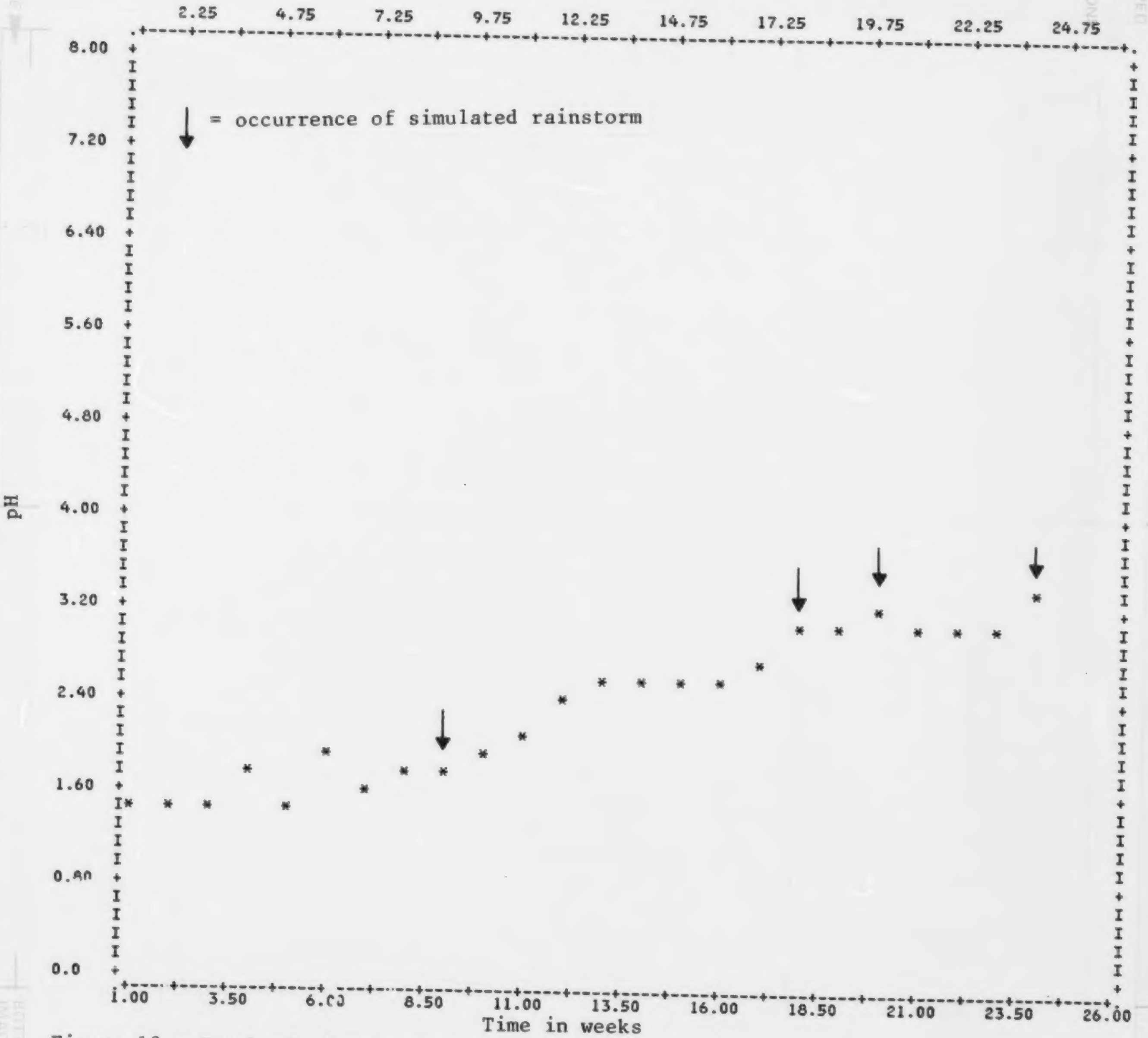

Figure 13. Graph of pH as a function of time for Column 6 (incorporated amendments).

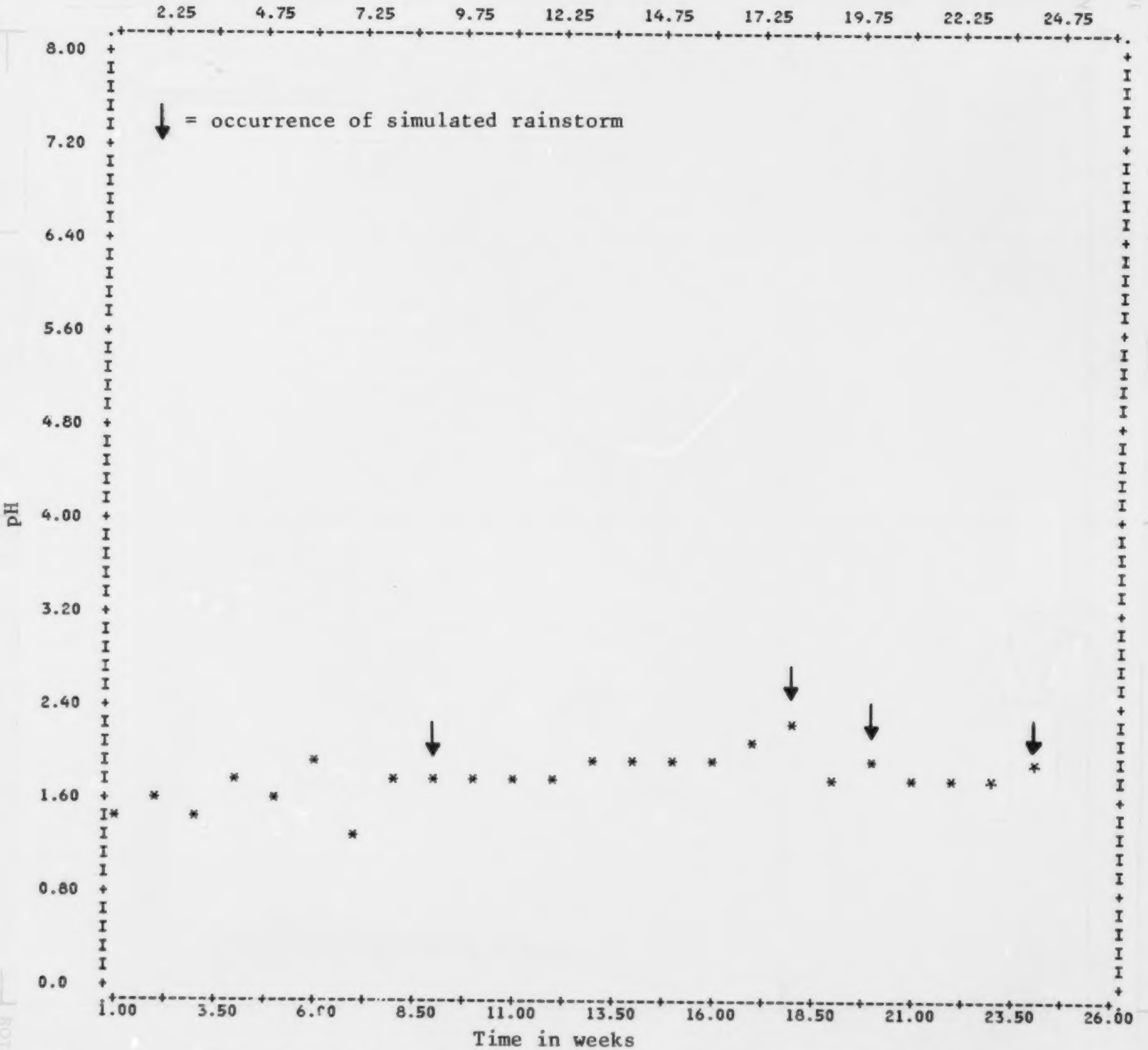

Figure 14. Graph of pH as a function of time for Column 7 (layered amendments).

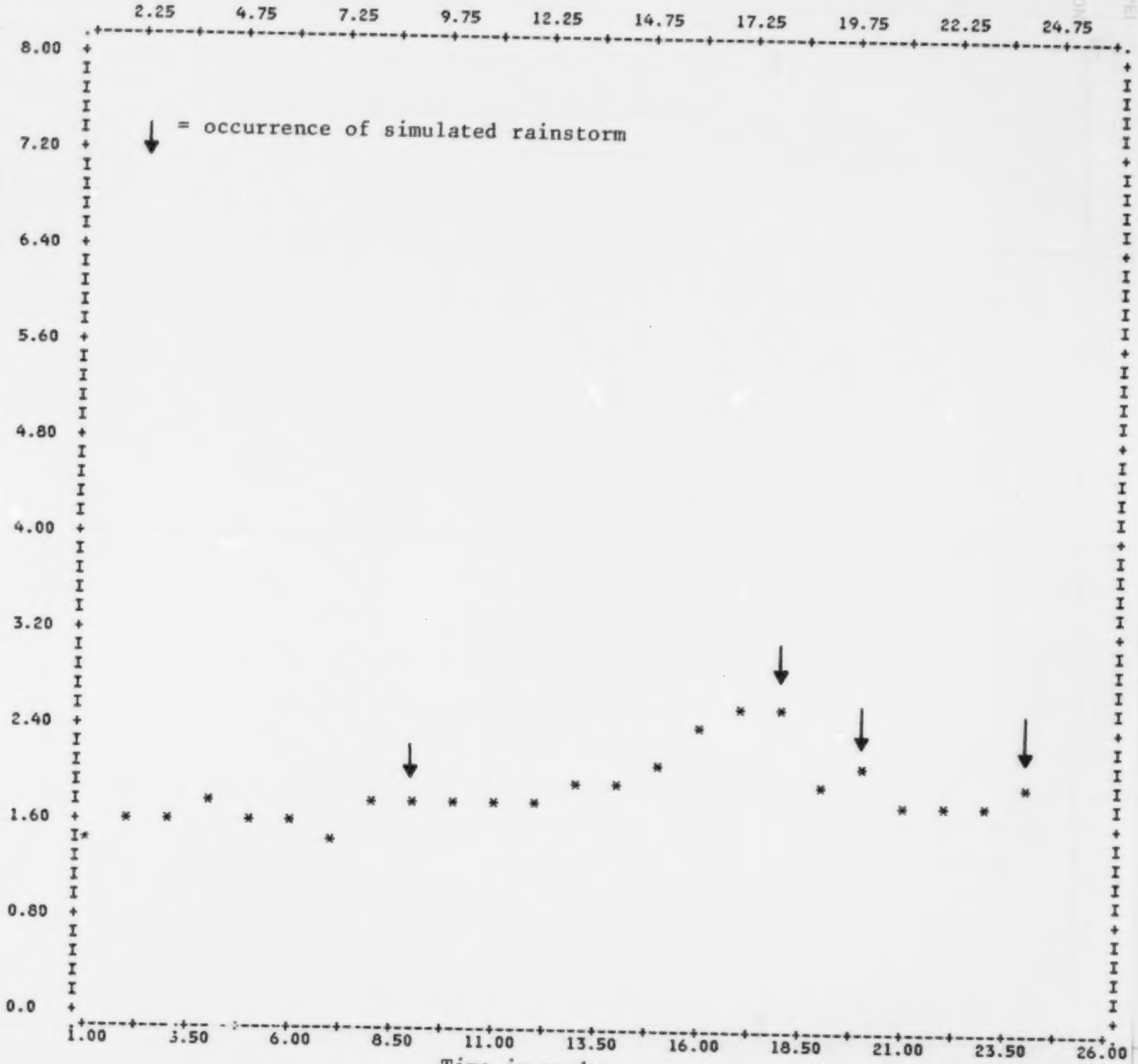

Figure 15. Graph of pH as a function of time for Column 8 (layered amendments).

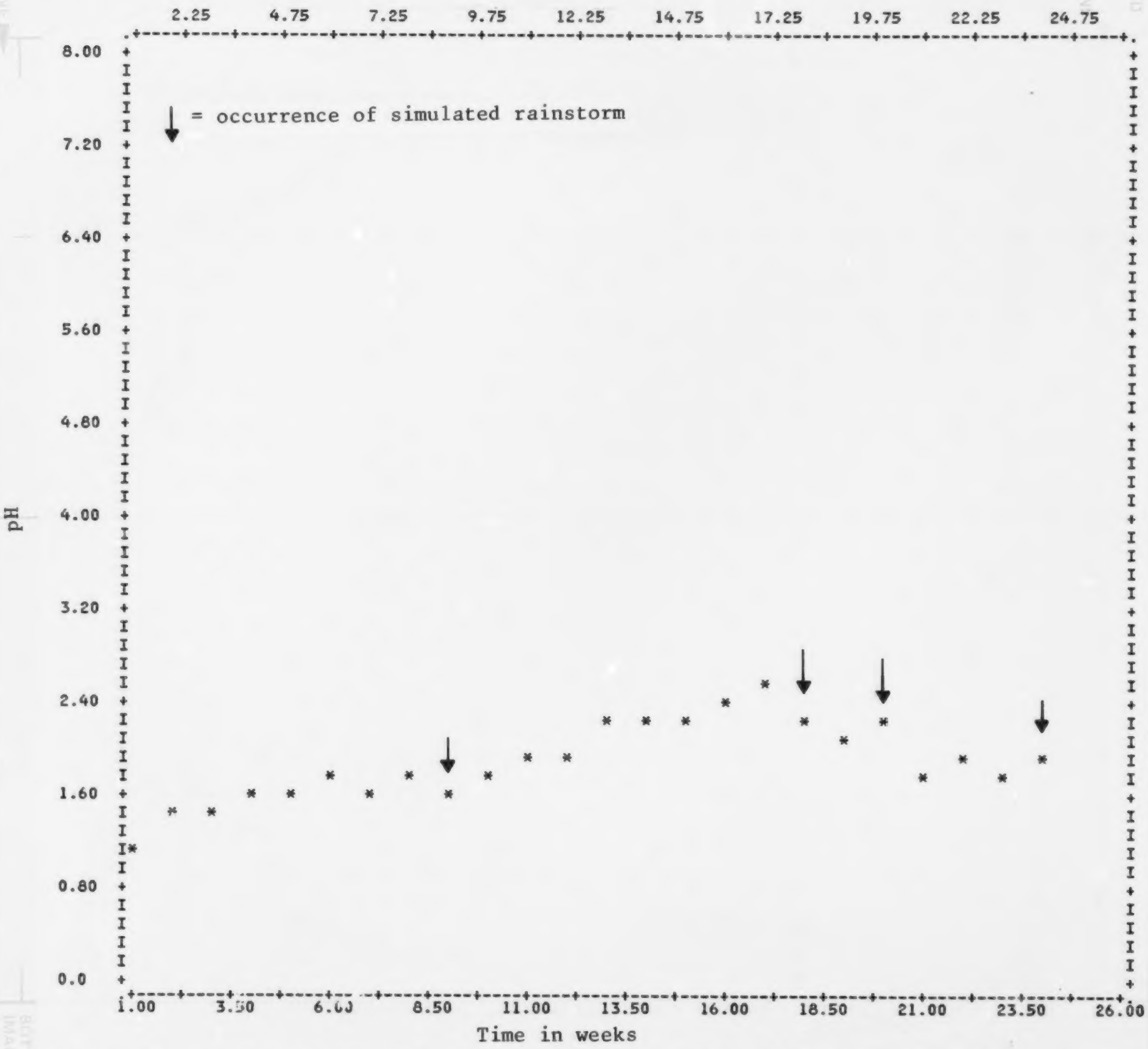

Figure 16. Graph of pH as a function of time for Column 9 (0.46-m (1.5-ft) cover soil layer).

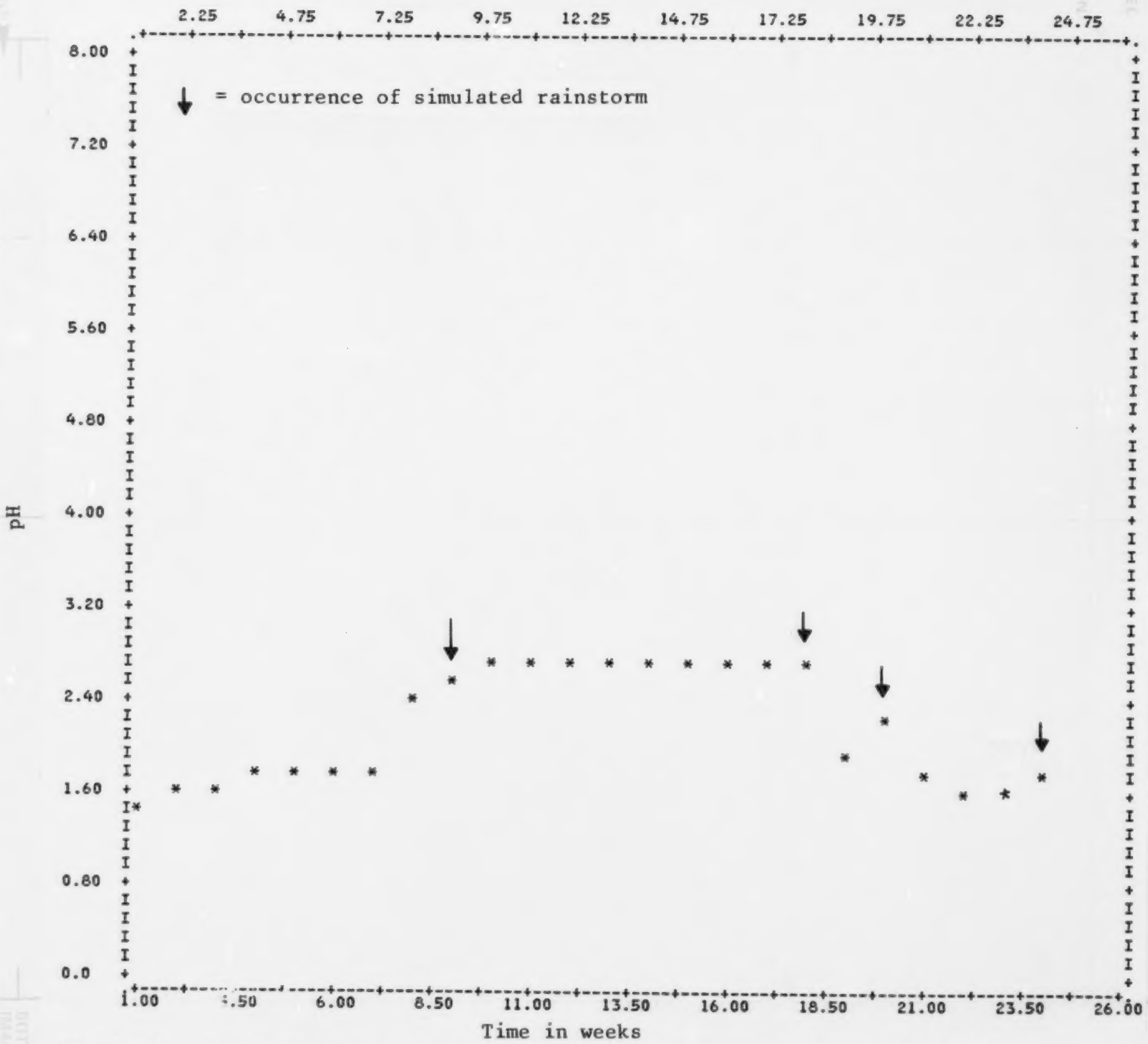

Figure 17. Graph of pH as a function of time for Column 10 (0.46-m (1.5-ft) cover soil layer).

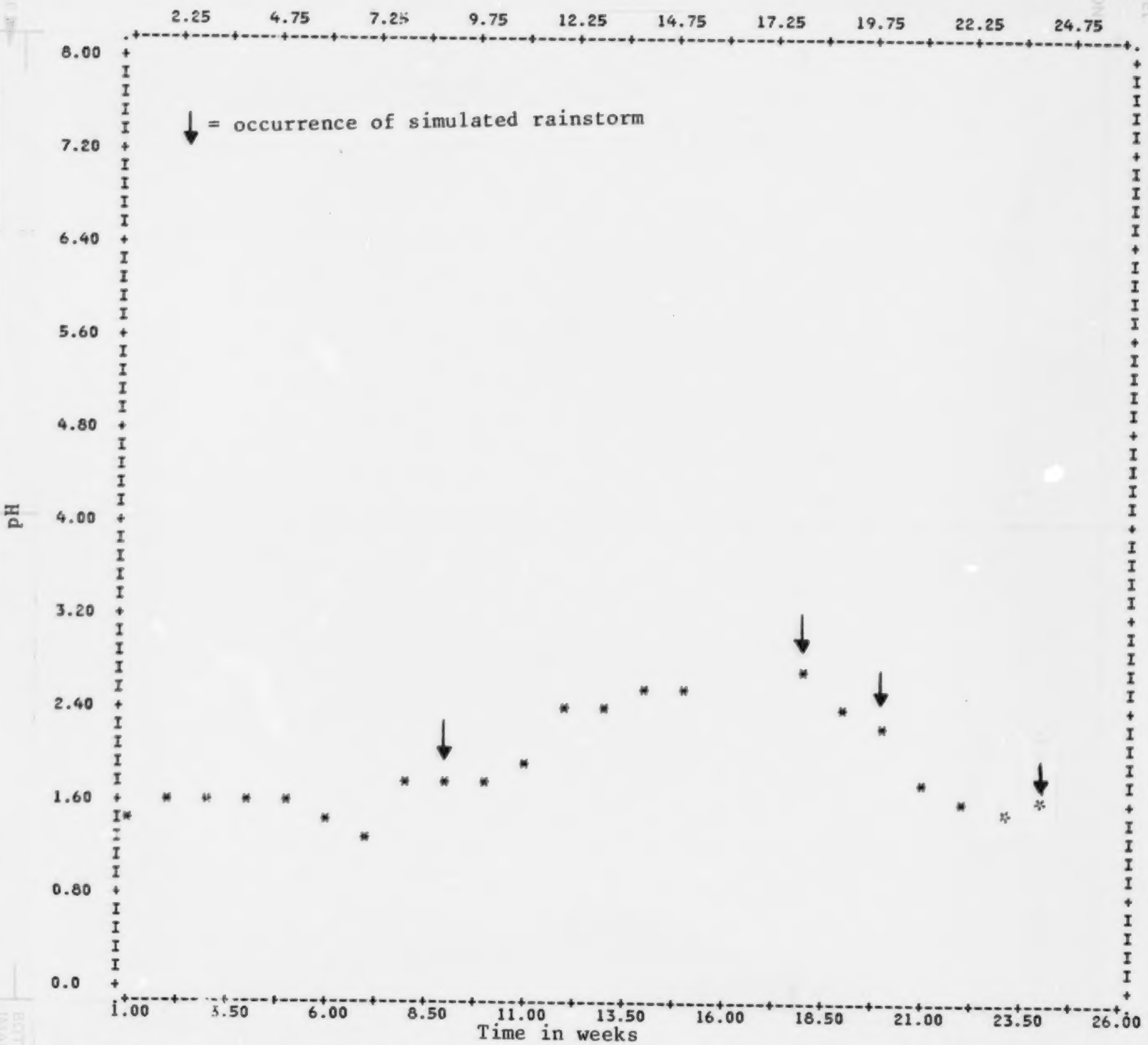

Figure 18. Graph of pH as a function of time for Column 11 (0.61-m (2-ft) cover soil layer).

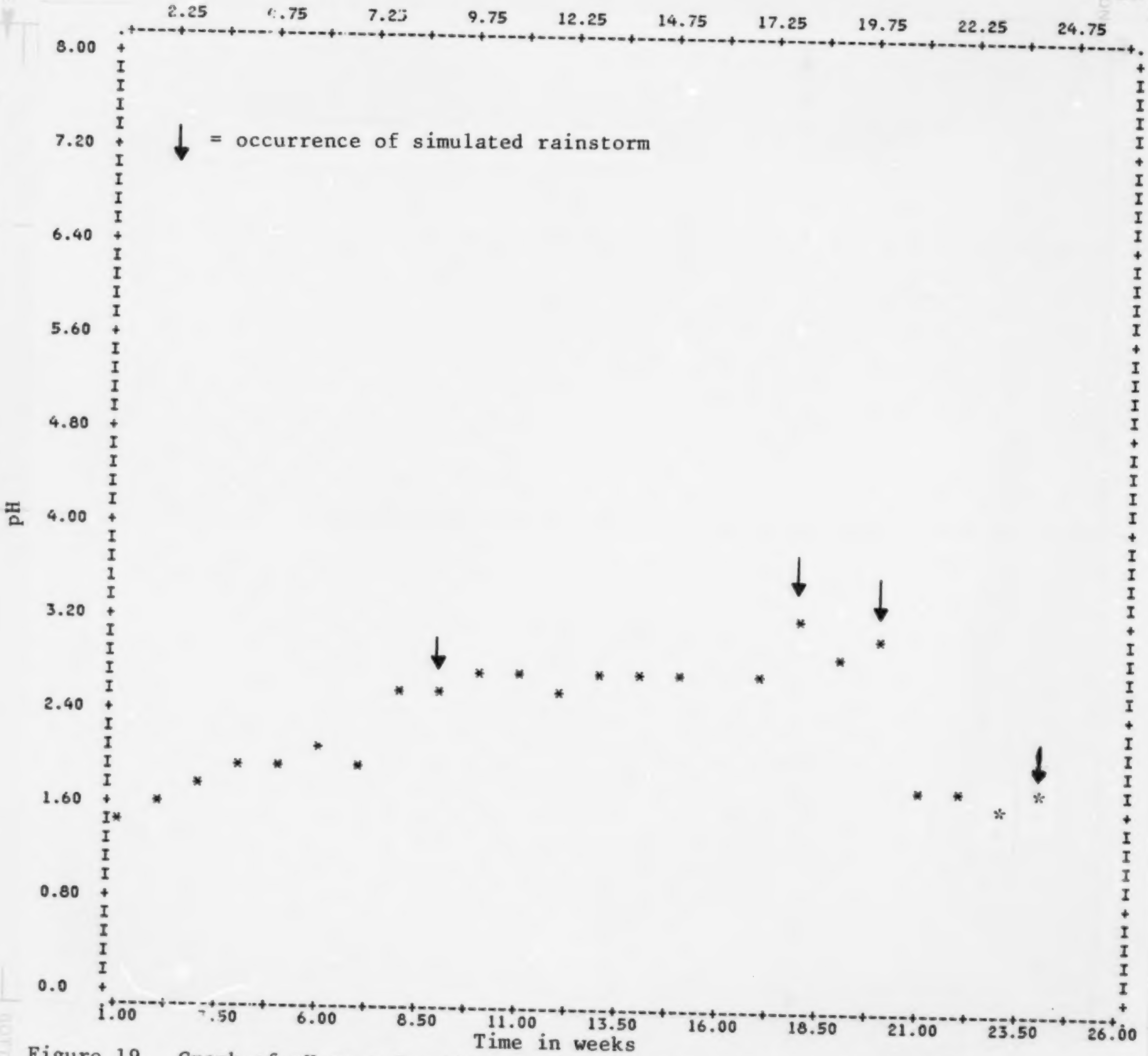

Figure 19. Graph of pH as a function of time for Column 12 (0.61-m (2-ft) cover soil layer).

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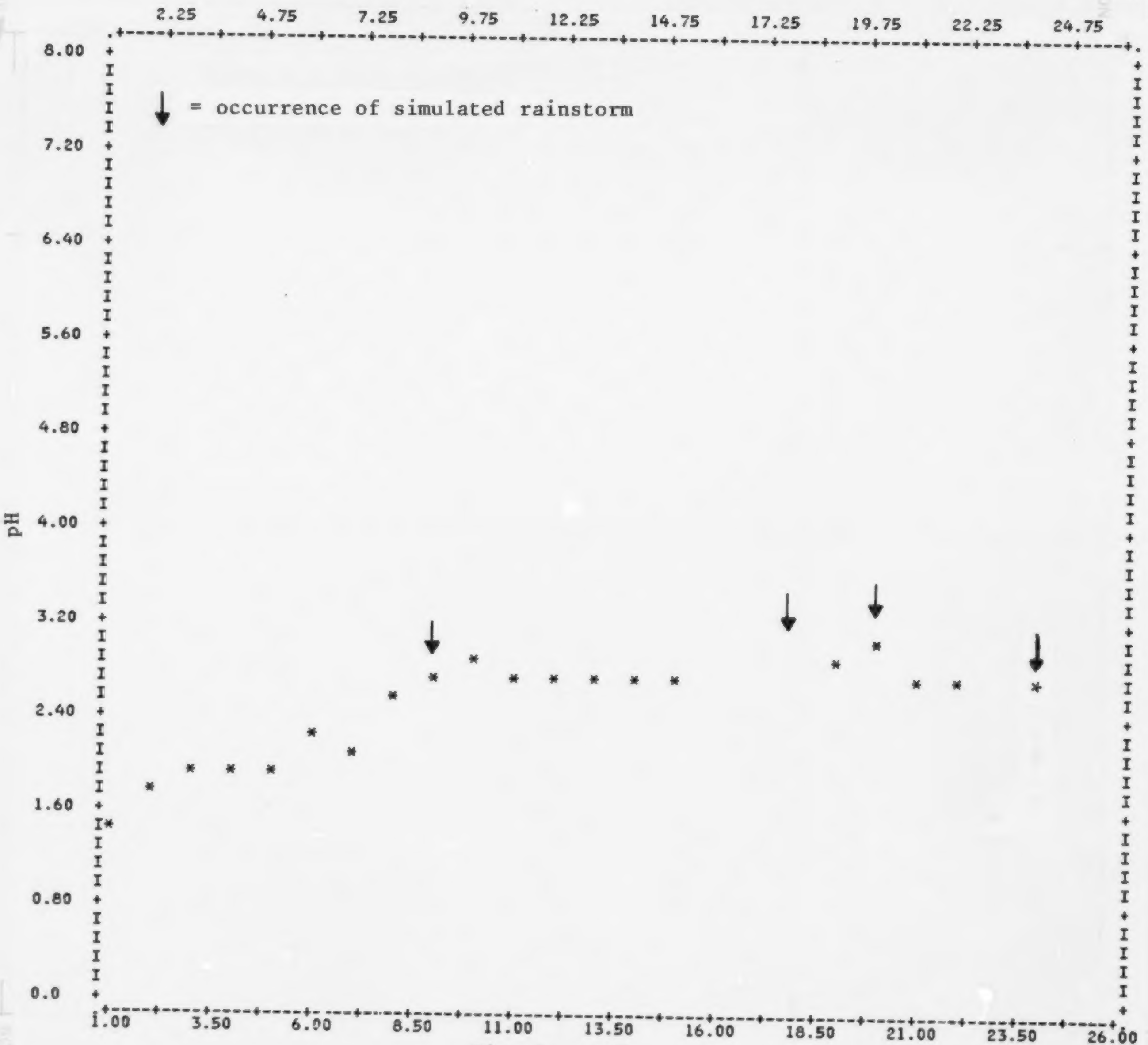

Figure 20. Graph of pH as a function of time for Column 13 (1.2-m (4-ft) cover soil layer).

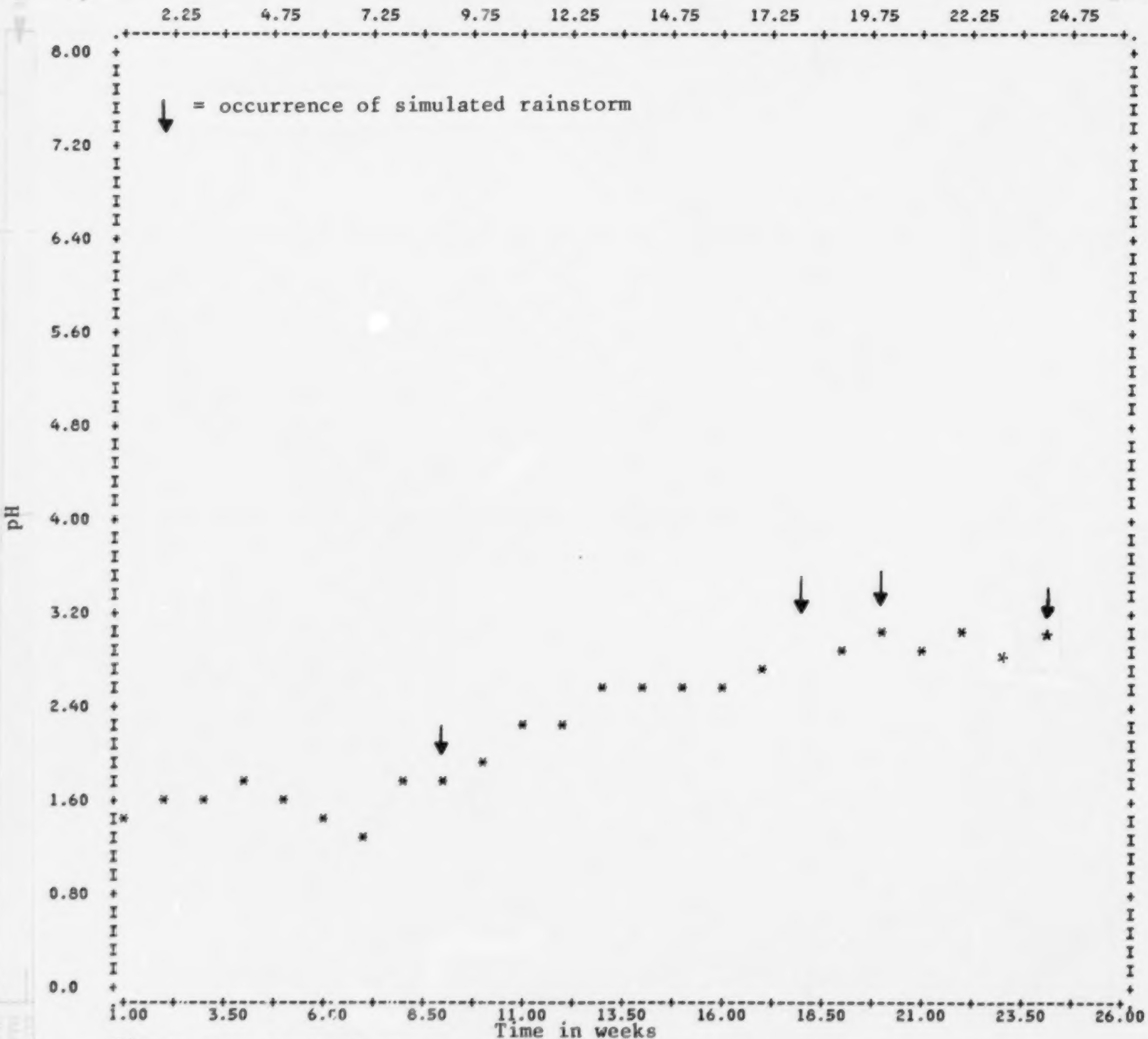

Figure 21. Graph of pH as a function of time for Column 14 (1.2-m (4-ft) cover soil layer).

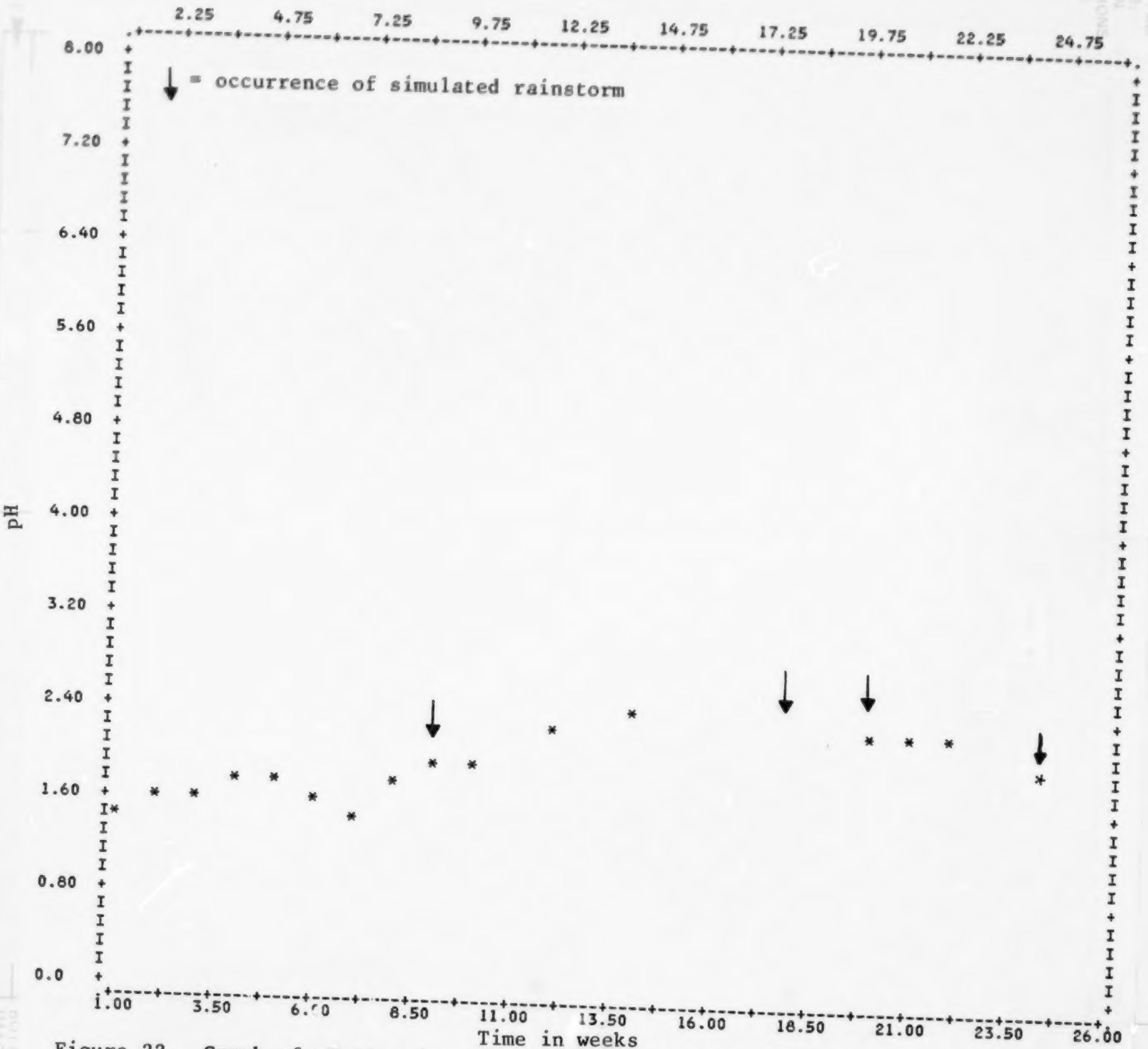

Figure 22. Graph of pH as a function of time for Column 15 (2.5-cm (1-in) neutralizing layer).

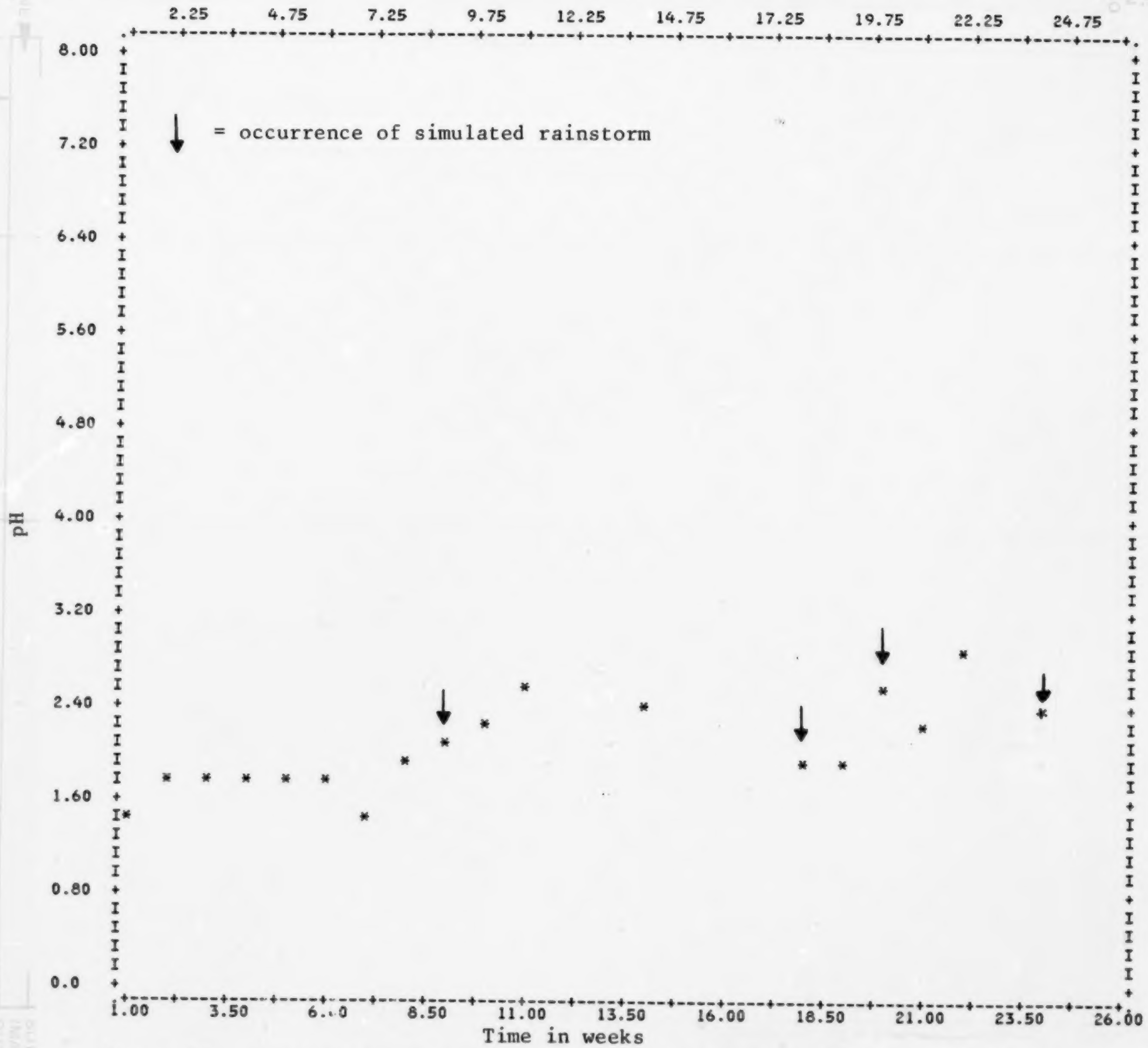

Figure 23. Graph of pH as a function of time for Column 16 (cover soil/mine waste control)

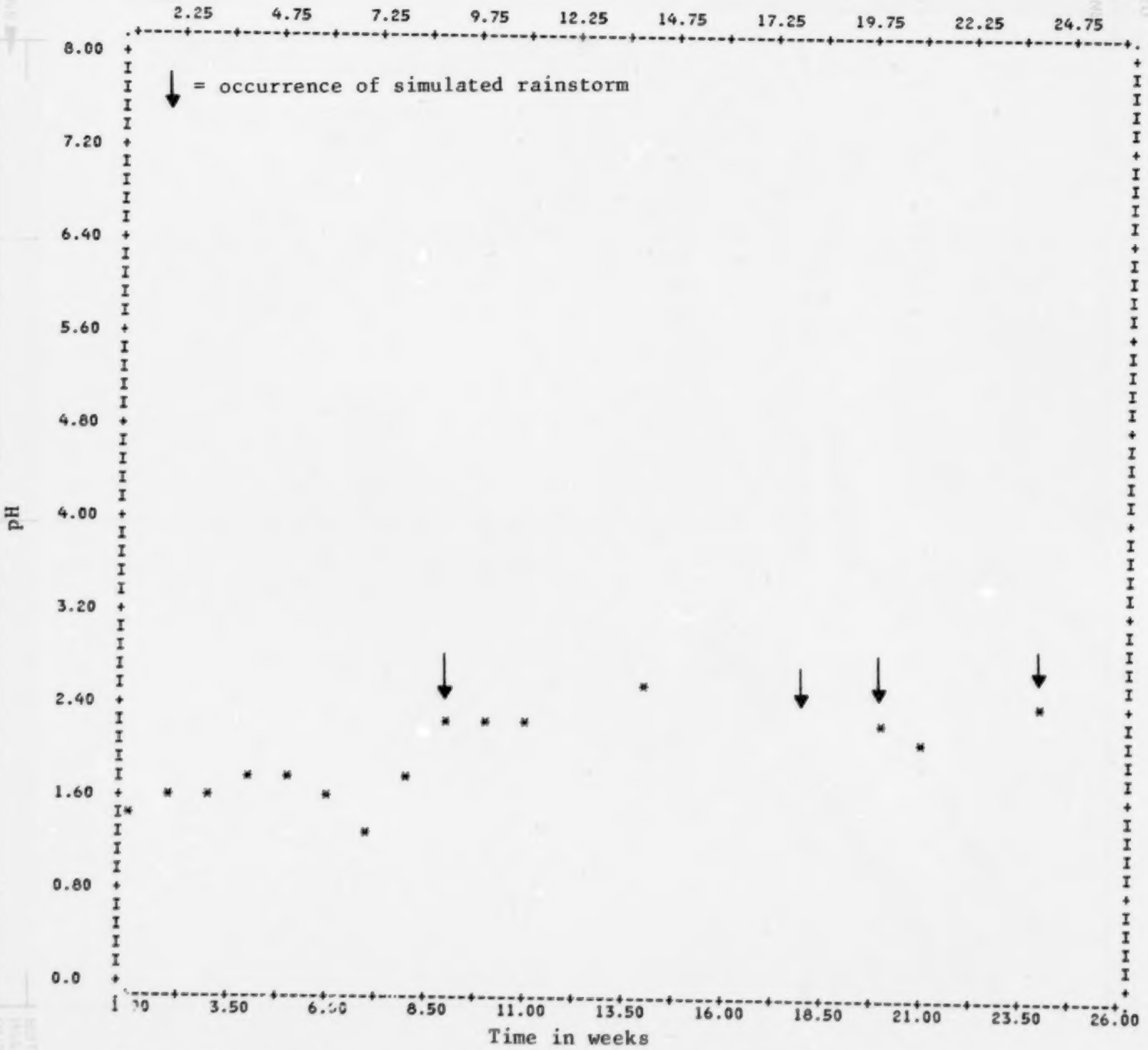

Figure 24. Graph of pH as a function of time for Column 17 (5.0-cm (2-in) neutralizing layer).

also true for columns containing 0.5 m (1.5 ft) of cover soil. These rates of change in pH over the study period for these columns were within one percent of those for the columns containing 0.61 m (2 ft) of cover soil. There was little difference in the pH of columns 9 through 12 as evidenced by Figures 16 through 19.

The pH of the leachates generated from columns containing a neutralizing layer positioned between the mine waste and cover soil material (columns 15 and 17) showed little variation over the study period from the control column (column 16) containing similar quantities of the mine waste and cover soil material (Figures 22 through 24). The leachate pH of the control column ranged from approximately 1.4 to 2.9. In comparison, leachate generated from column 15, containing approximately 1.6 kg (3.5 lbs) of agricultural limestone (approximately a 2.5-cm (1-in) layer), ranged from 1.4 to 2.4. Similarly, the leachate pH of column 17, containing 3.2 kg (7.0 lbs) of limestone (approximately a 5.0 cm (2 in) layer), ranged from 1.4 to 2.5 as shown in Figure 24.

Acidity Data

Leachate acidity closely parallelled leachate pH throughout the study period. The acidity raw data was recorded in mg/l (see Appendix B) and later converted to mean load data using the statistical procedures previously described. The mean acidity load data for each pair of columns was then plotted as a function of time as shown in Figures 25 through 34.

The mean acidity loads of the leachate from the untreated pyrite waste control columns (columns 1 and 2) ranged from approximately 44 g/day to over 500 g/day, averaging approximately 172 g/day (Figure 25). Maximum acidity loads correlated with increased amounts of water applied to the columns to simulate 10-year, 24-hour storm events at weeks 9, 18, 20 and 24 of the study period. Leachate mean acidity loads from the soil control columns (columns 3 and 4) were low (Figure 26) and considered typical for a normal top soil.

In comparison to the mine waste control columns (columns 1 and 2) the incorporation of lime, fertilizer and sludge within the top 15.2 cm (6 in) of the pyritic waste (columns 5 and 6) produced a dramatic effect on leachate acidity as shown in Figure 27. The mean acidity load of the leachate from these columns ranged from approximately 2 to 200 g/day and averaged approximately 41 g/day. An overall 77-percent decrease in the mean acidity load was realized when the amendments were incorporated into the upper strata of the mine waste (column 5 and 6). This compared to an overall decrease of 65 percent in the mean acidity load of the leachate resulting from columns 7 and 8 which contained the sludge and fertilizer layered over the lime-treated mine waste. The leachate mean acidity load from these columns ranged from less than 1 g/day to over 260 g/day (Figure 28) and averaged approximately 60 g/day. The rate of increase in the leachate mean acidity load resulting from the columns containing the materials incorporated (columns 5 and 6) was approximately nine times greater than the leachate mean acidity load from the columns containing the materials layered into the upper strata of the mine waste (columns 7 and 8).

The mean acidity load of the leachate generated from columns containing

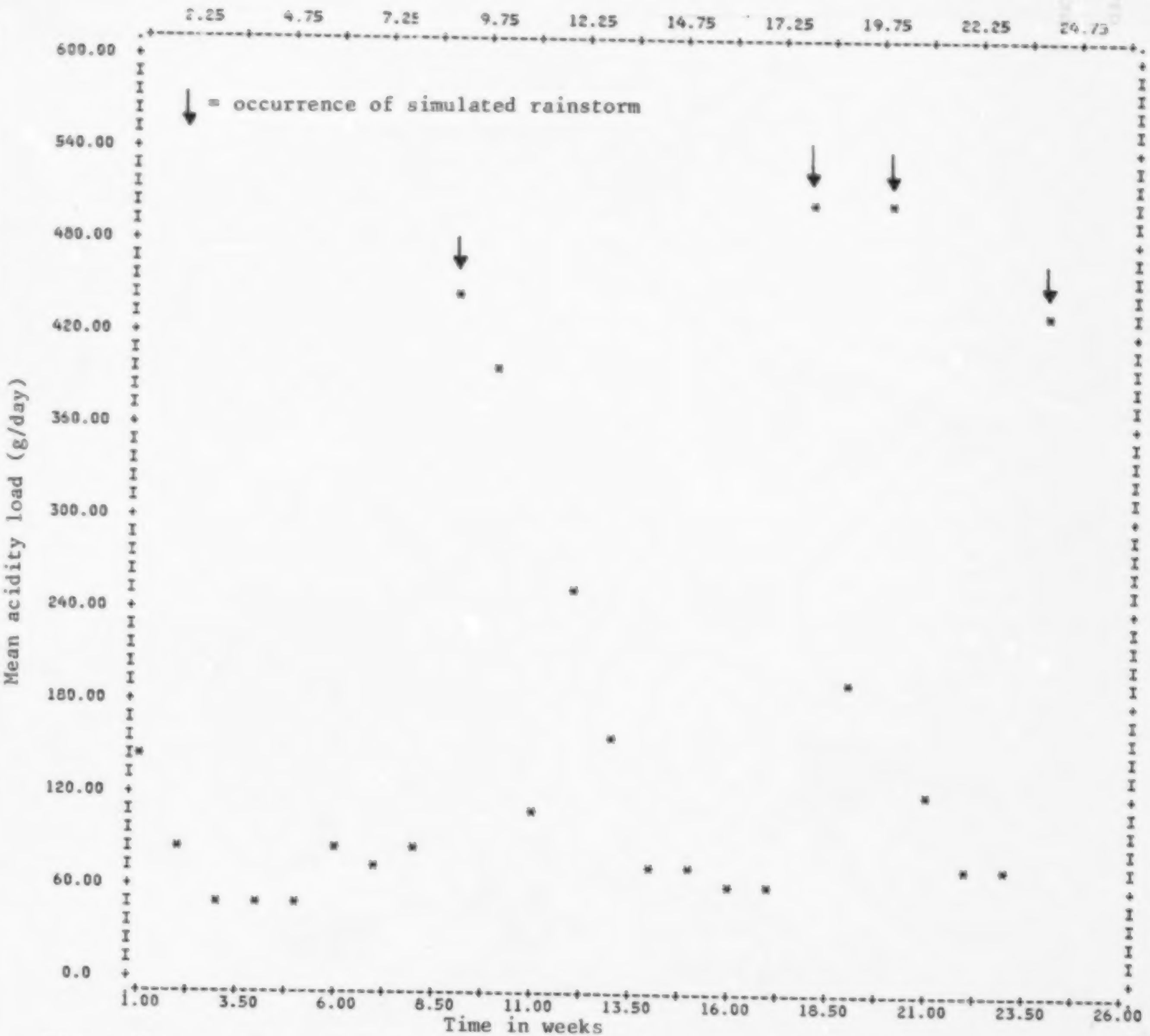

Figure 25. Graph of mean acidity load as a function of time for Columns 1 and 2 (mine waste controls).

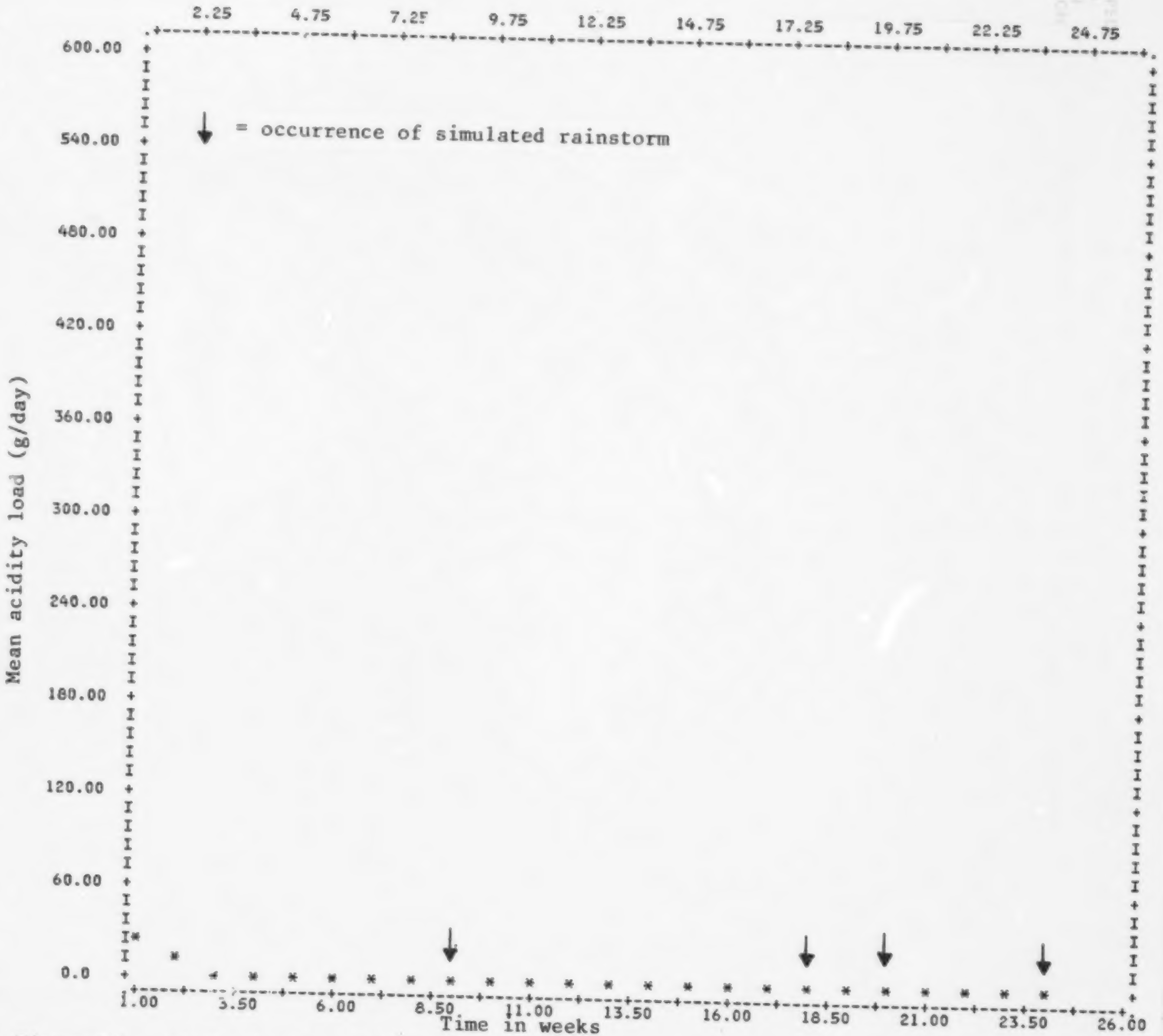

Figure 26. Graph of mean acidity load as a function of time for Columns 3 and 4 (cover soil controls).

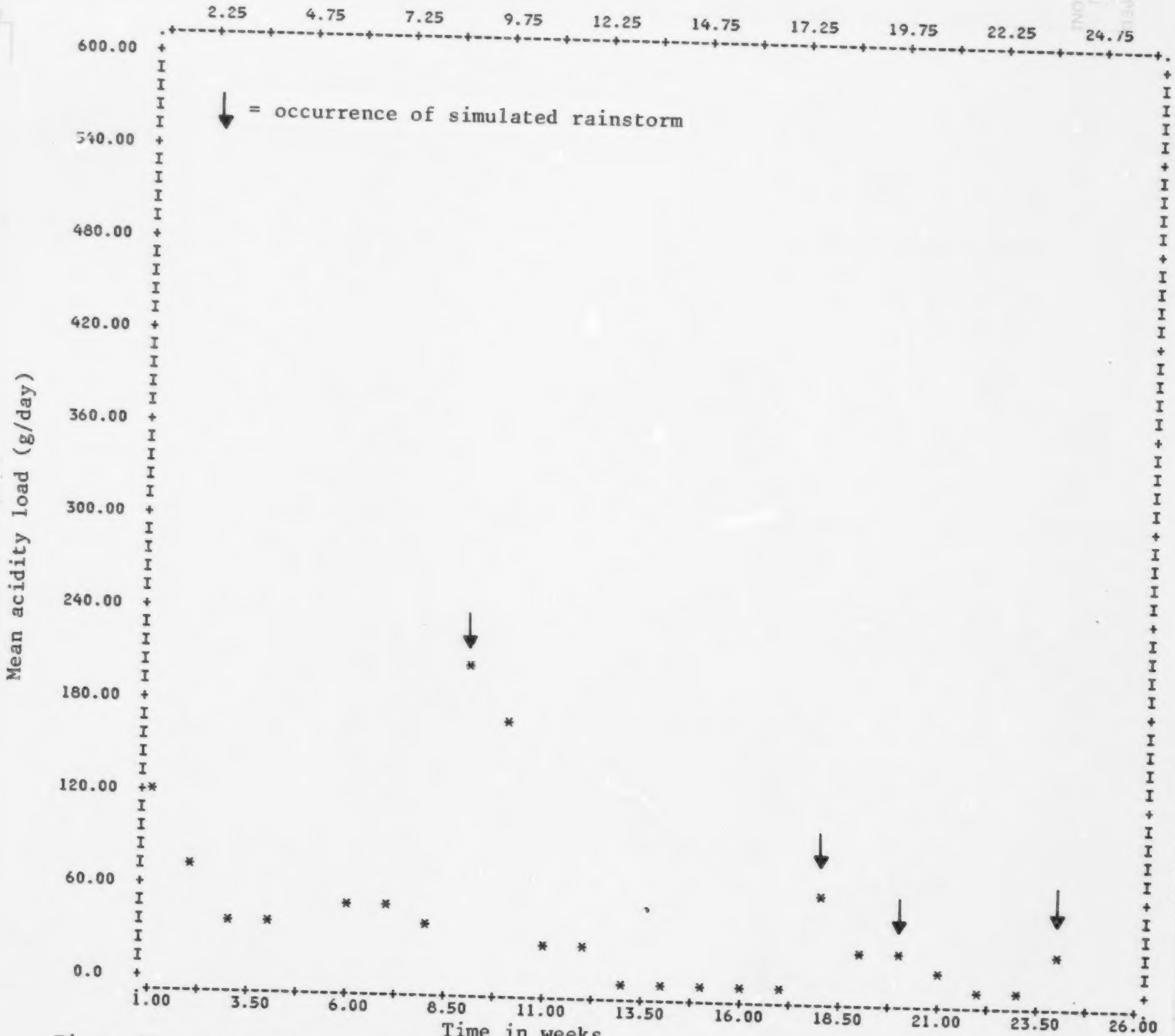

Figure 27. Graph of mean acidity load as a function of time for Columns 5 and 6 (incorporated amendments).

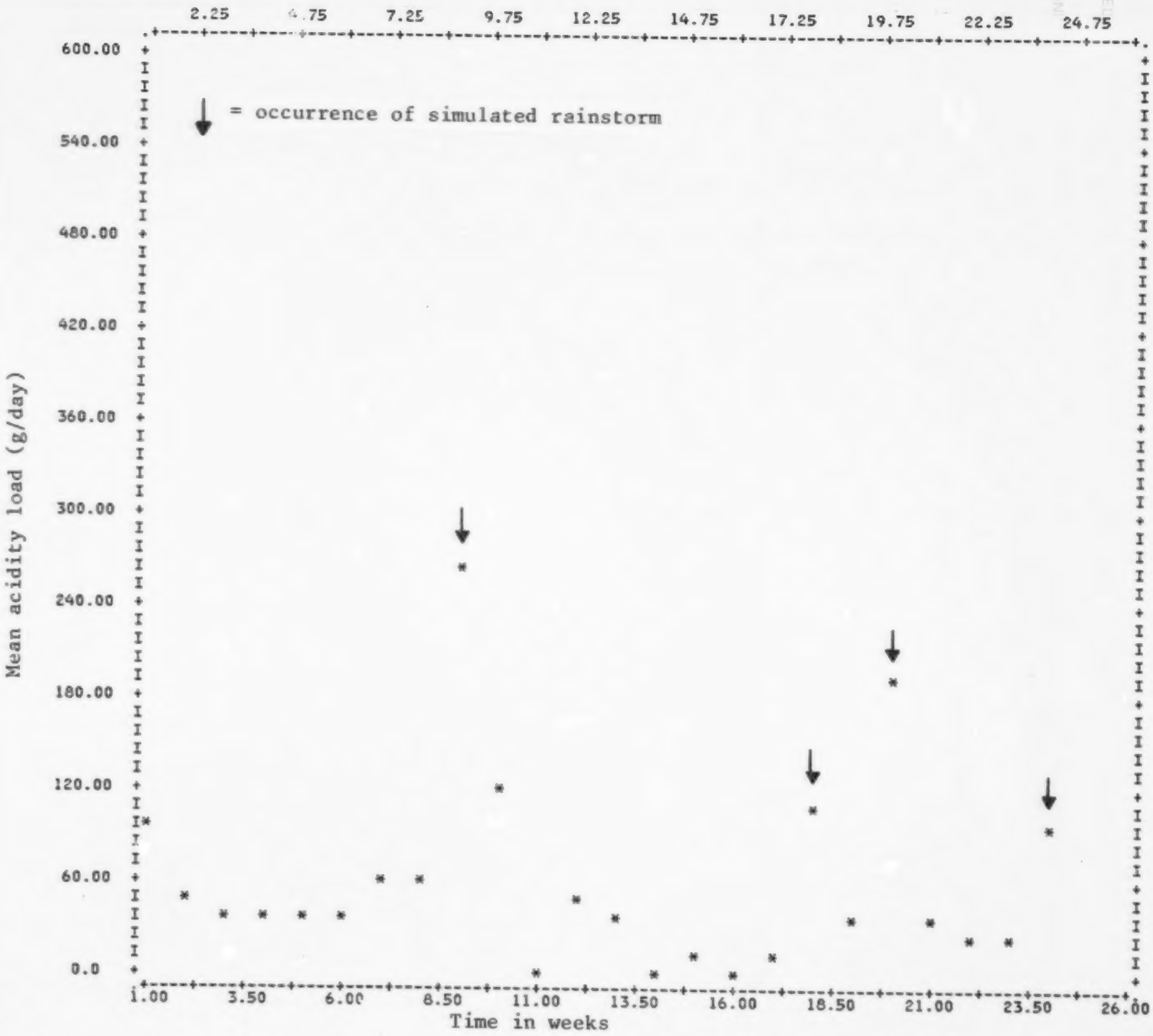

Figure 28. Graph of mean acidity load as a function of time for Columns 7 and 8 (layered amendments).

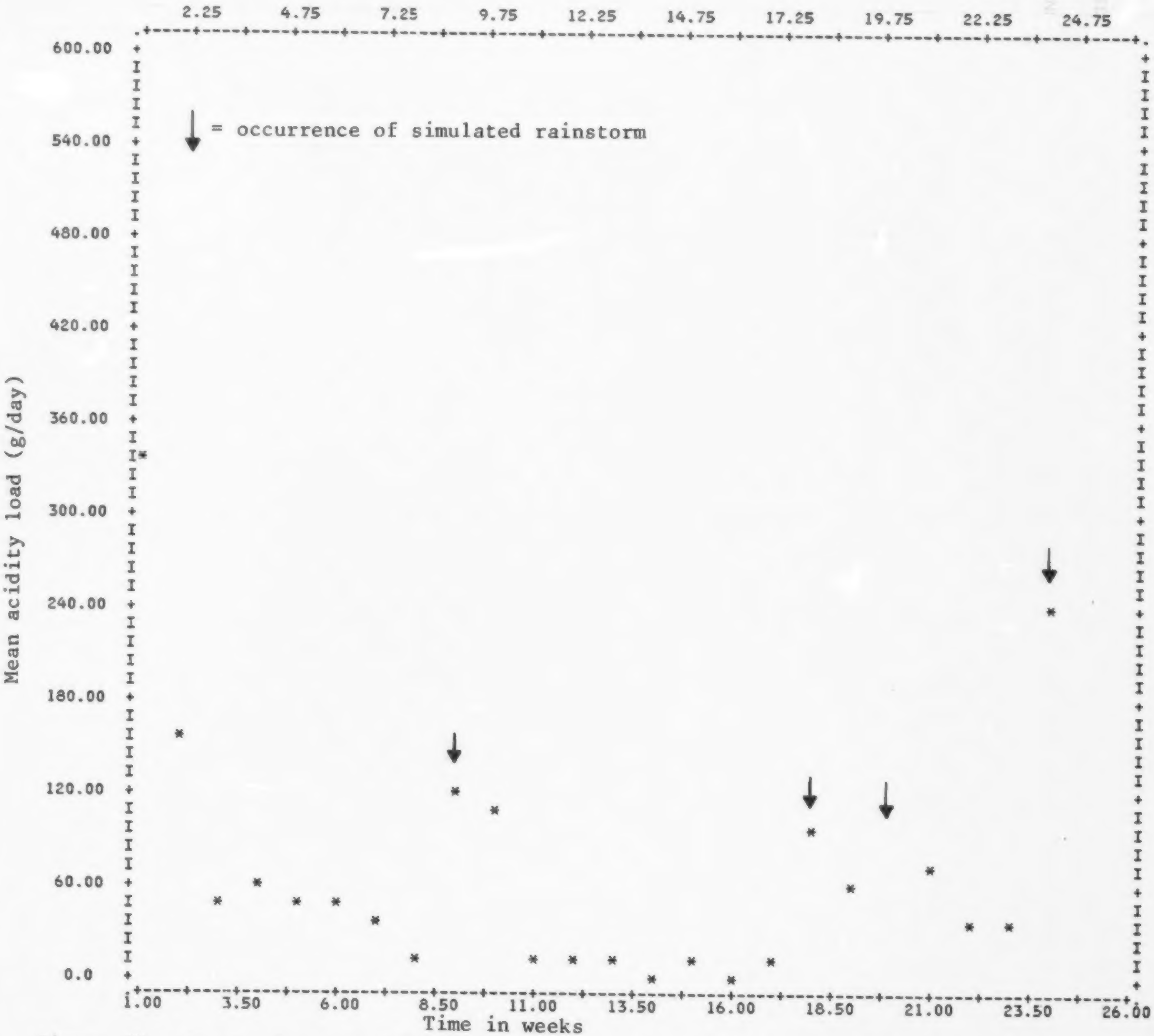

Figure 29. Graph of mean acidity load as a function of time for Columns 9 and 10 (0.46-m (1.5-ft) cover soil l

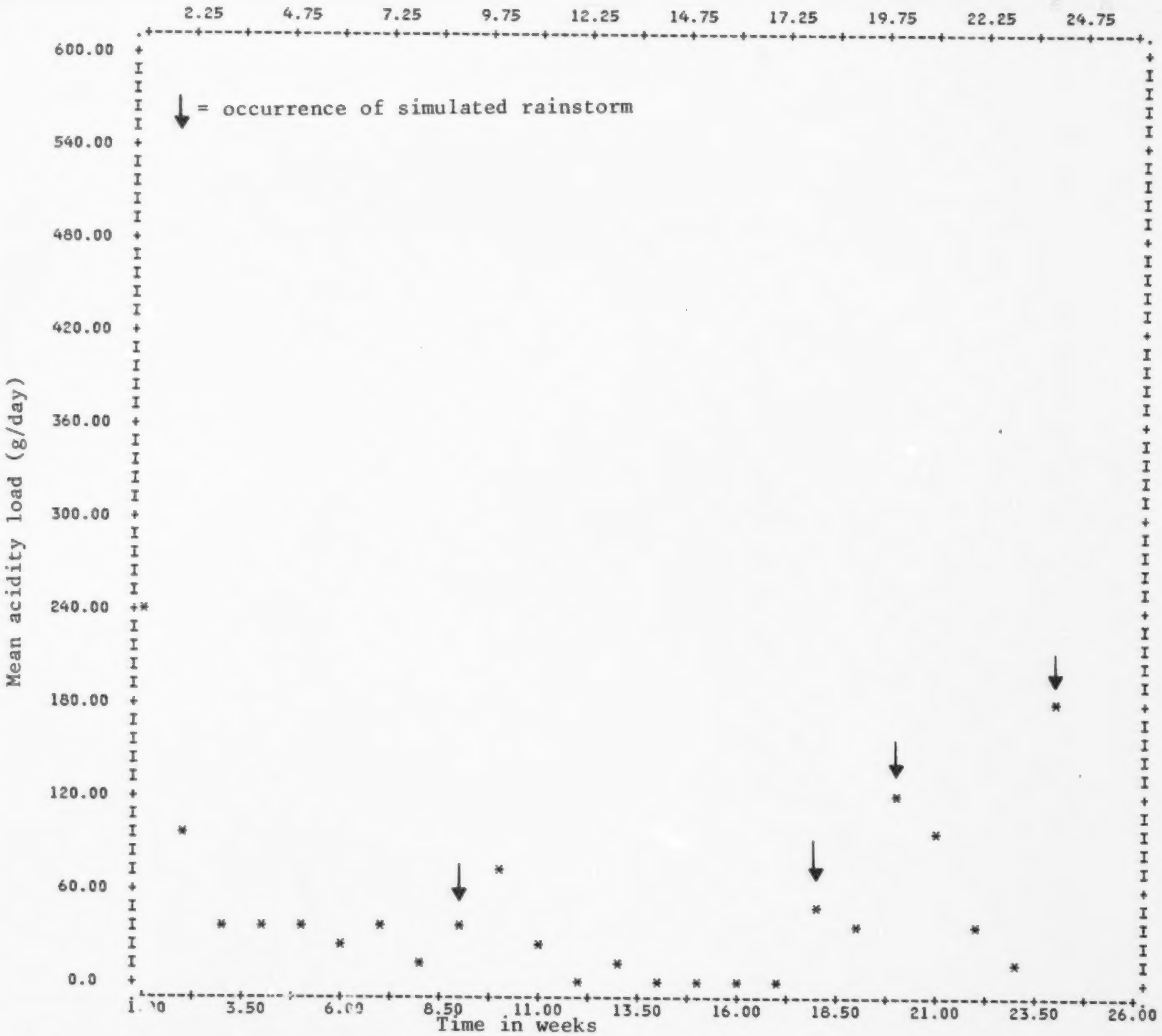

Figure 30. Graph of mean acidity load as a function of time for Columns 11 and 12 (0.61-m (2-ft) cover soil layer).

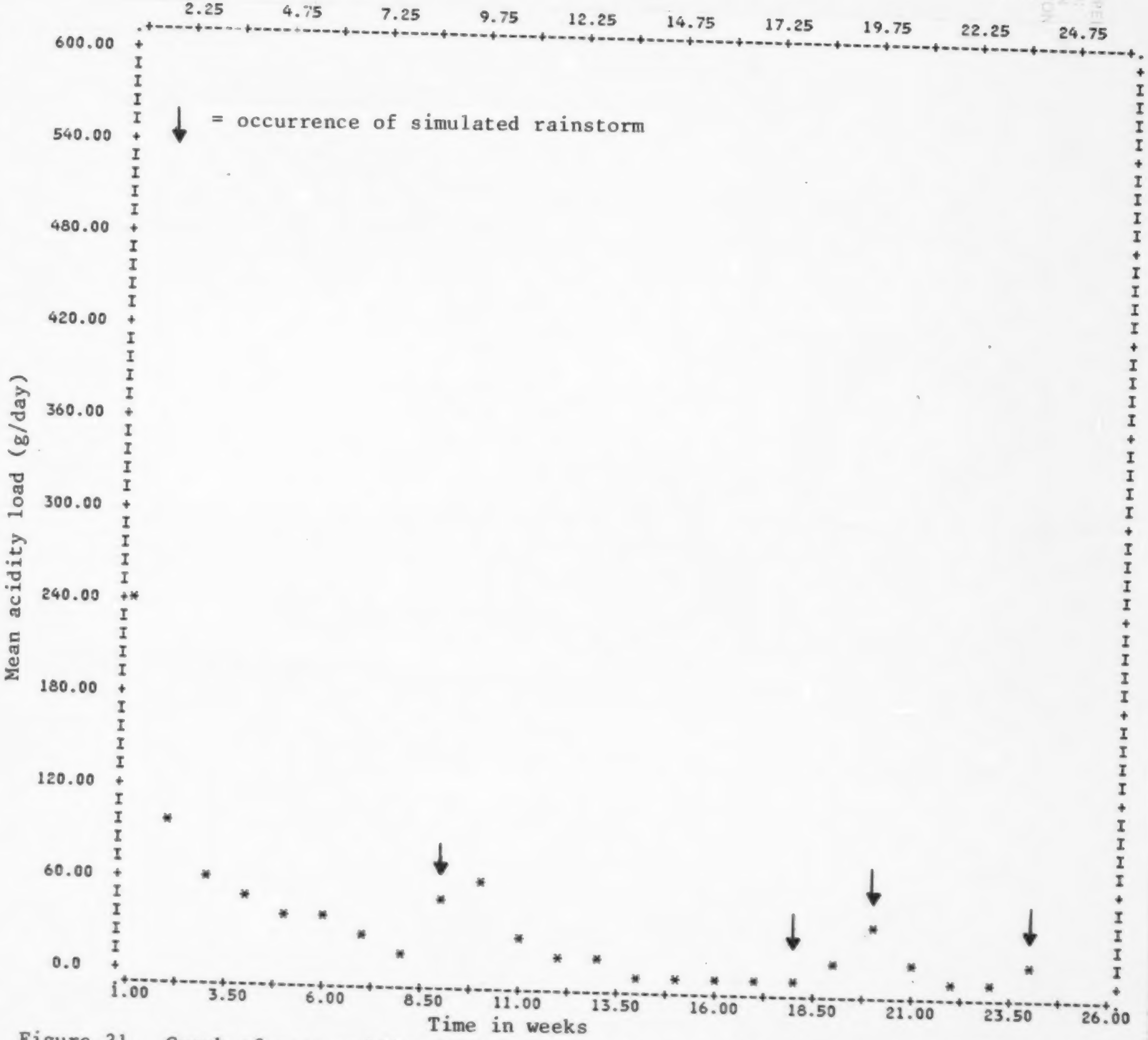

Figure 31. Graph of mean acidity load as a function of time for Columns 13 and 14 (1.2-m (4-ft) cover soil layer).

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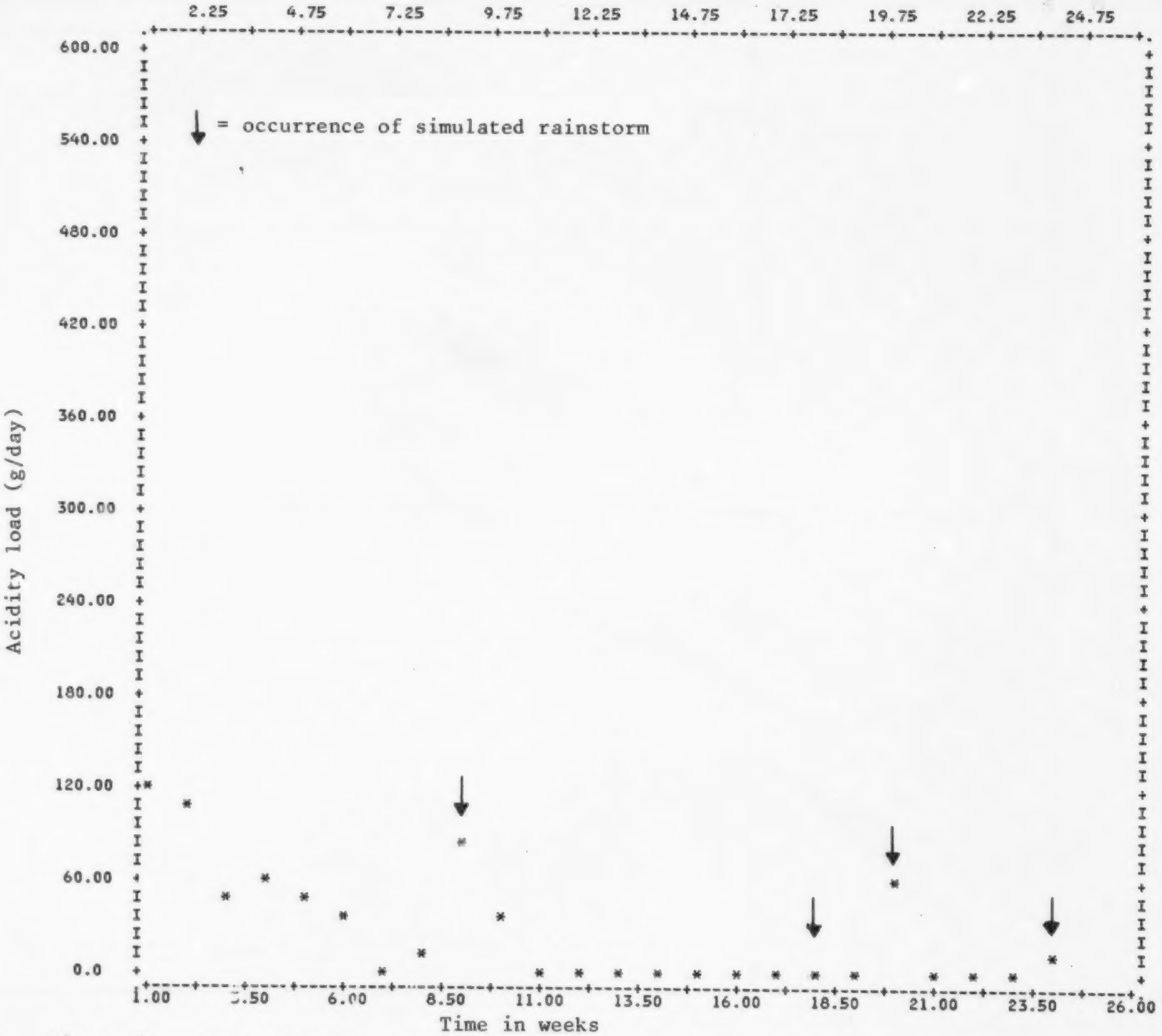

Figure 32. Graph of acidity load as a function of time for Column 15 (2.5-cm (1-in) neutralizing layer).

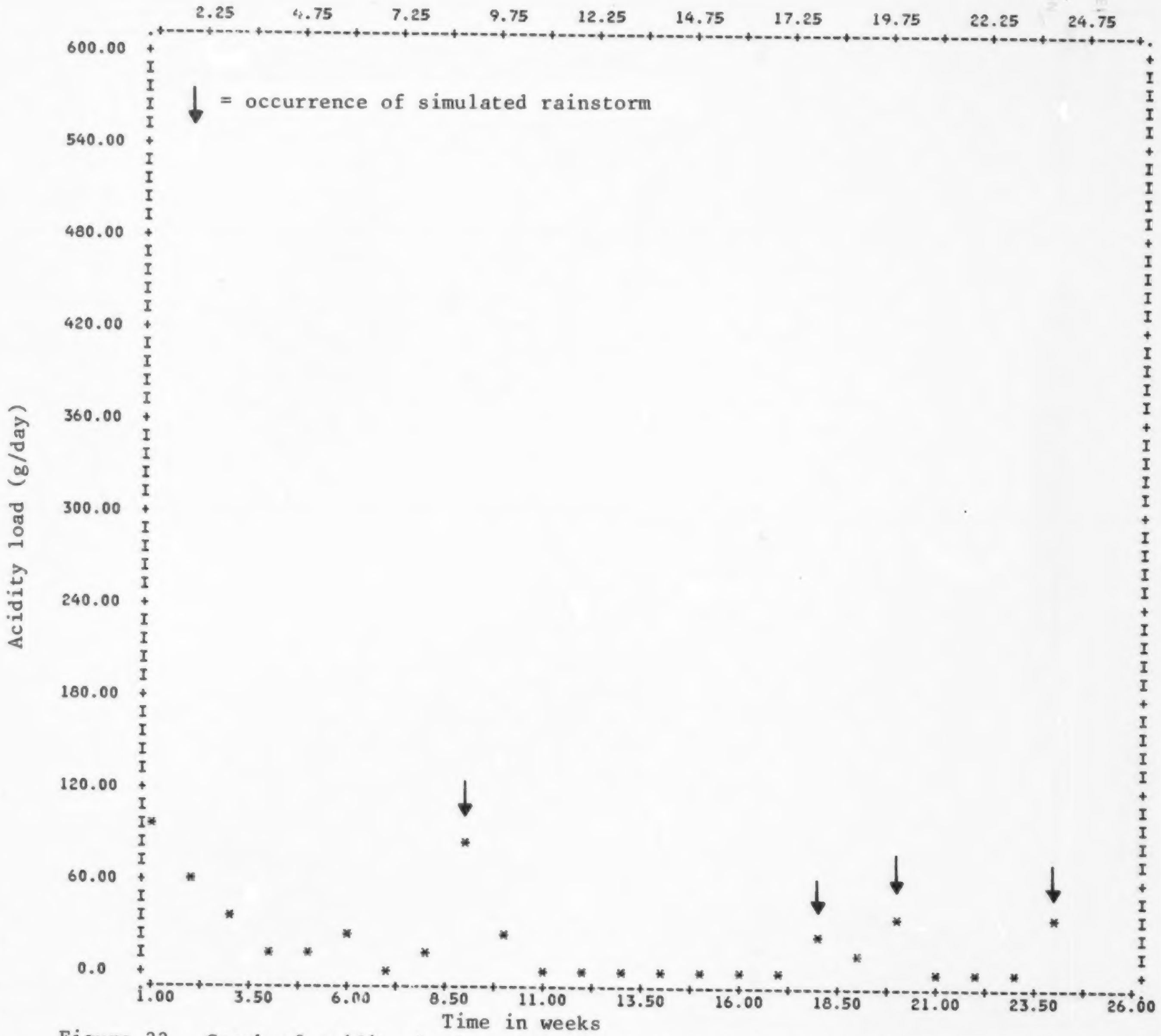

Figure 33. Graph of acidity load as a function of time for Column 16 (cover soil/mine waste control).

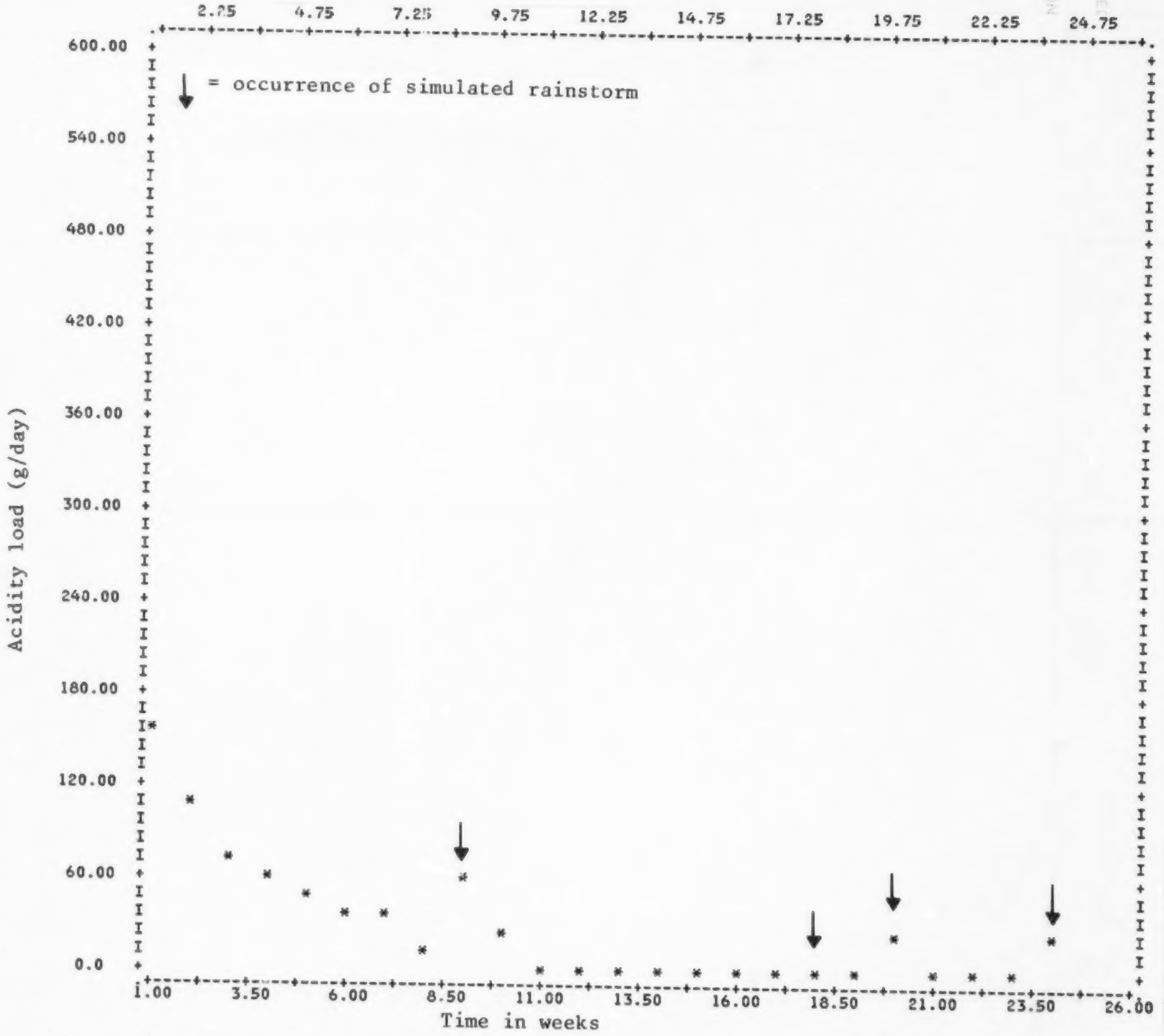

Figure 34. Graph of acidity load as a function of time for Column 17 (5.0-cm (2-in) neutralizing layer).

the pyrite mine waste covered with 4.6 m (1.5 ft) of cover soil (columns 9 and 10) decreased approximately 58 percent compared to that from the columns containing the untreated pyrite mine waste (columns 1 and 2). The leachate mean acidity loads resulting from these columns ranged from an initial maximum load of approximately 340 g/day to a minimum load of approximately 5 g/day recorded at week 14 (Figure 29). The mean acidity loads of this leachate averaged around 70 g/day. In comparison, placement of a 0.61-m (2-ft) layer of cover soil over the mine waste (columns 11 and 12) resulted in an average leachate mean acidity load of approximately 50 g/day and an overall decrease of 71 percent in the mean acidity load of this leachate when compared to that of the control columns (columns 1 and 2). Mean acidity loads in the leachate from columns 11 and 12 ranged from 242 g/day to less than 1 g/day (Figure 30). As in the case of pH, little difference in acidity load was noted between the 0.5 m (1.5 ft) (Figure 29) and the 0.61 m (2 ft) (Figure 30) cover soil treatments. A more dramatic effect on the leachate acidity load, however, was noted when 1.2 m (4 ft) of cover soil (columns 13 and 14) were placed over the mine waste. The mean acidity loads from these columns decreased steadily (Figure 31) throughout the course of the experiment and resulted in an overall decrease of 81 percent when compared to the pyrite waste control columns with no cover soil (columns 1 and 2). This corresponded with the steady increase in pH (Figures 6 and 7) discussed earlier. The leachate mean acidity loads from columns 13 and 14 ranged from an initial maximum load of approximately 240 g/day to a minimum load of less than 1 g/day and averaged approximately 30 g/day. The rate of decrease in the leachate mean acidity loads from the columns containing 0.61 m (4 ft) of cover soil was approximately two and one half times that of the columns containing only 0.5 m (1.5 ft) of cover soil.

The leachate acidity load from column 16 (note: columns 15, 16, and 17 were not duplicated, therefore references are to leachate acidity loads and not leachate mean acidity loads), containing 0.61 m (2 ft) of cover soil placed over 0.61 m (2 ft) of the mine waste, ranged from approximately 100 g/day to less than 1 g/day and averaged around 19 g/day (Figure 33). Correspondingly, the leachate acidity load resulting from placement of a 2.5-cm (1-in) layer of lime between the mine waste and cover soil (column 15) ranged from 120 g/day to also less than 1 g/day. The same was true when a 5-cm (2-in) lime layer (column 17) was used (Figure 34). Overall, few differences were noted among the plots of the acidity loads from these columns over the course of the study period. The only significant difference observed was in the rate of decrease in acidity load over time. Use of a 25-cm (1-in) layer of lime resulted in a rate of decrease in leachate acidity load approximately 1.8 times that of the control (column 16). Likewise, use of a 50-cm (2-in) layer of lime resulted in a rate of decrease approximately 2.3 times that of the same control.

COLUMN LEACHATE SPECIFIC CONDUCTANCE AND METAL DATA

Specific Conductance

Figures 35 through 51 illustrate the specific conductance of the column leachate as a function of time for the duration of the study period. Overall,

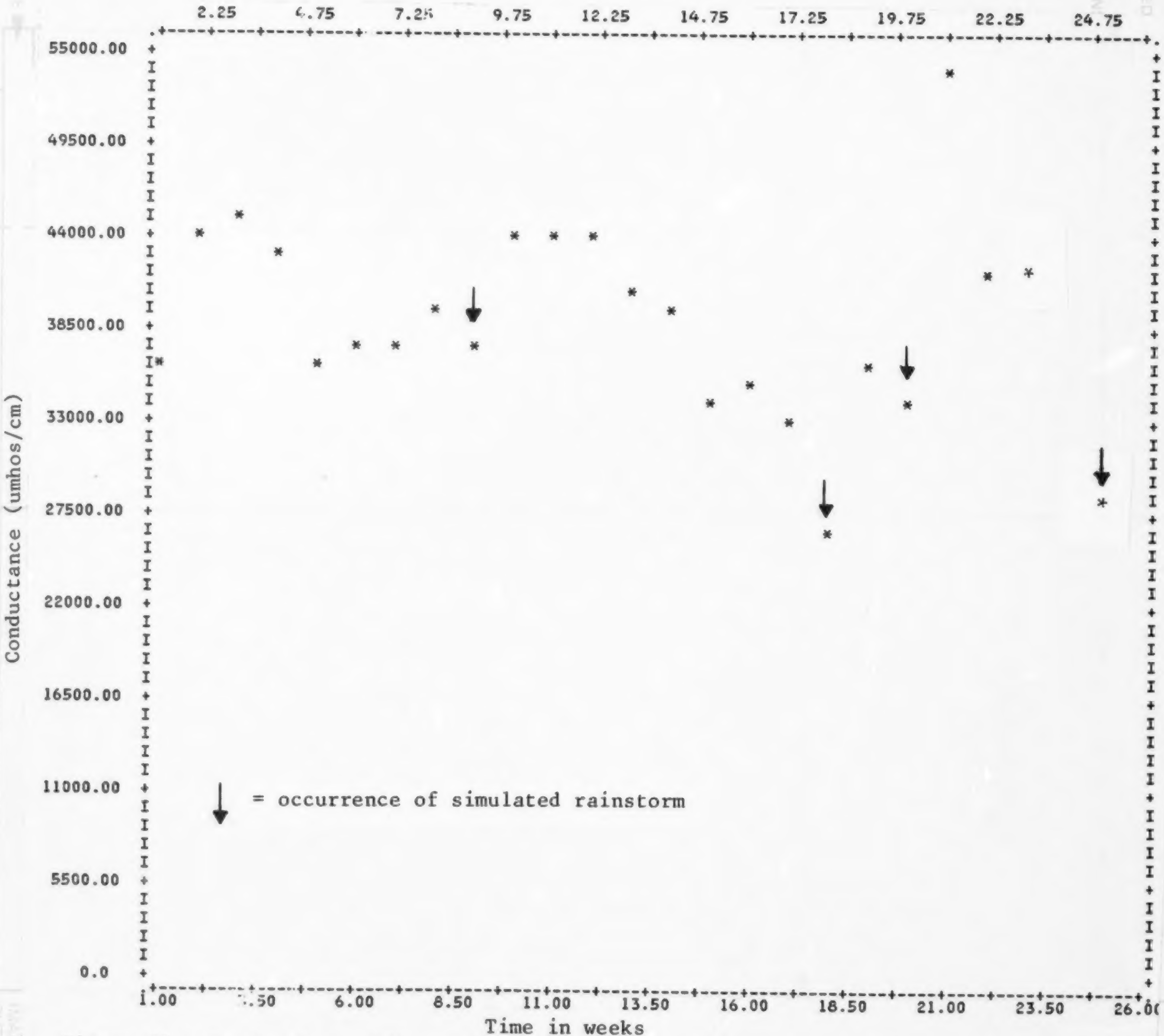

Figure 35. Graph of specific conductance as a function of time for Column 1 (mine waste control).

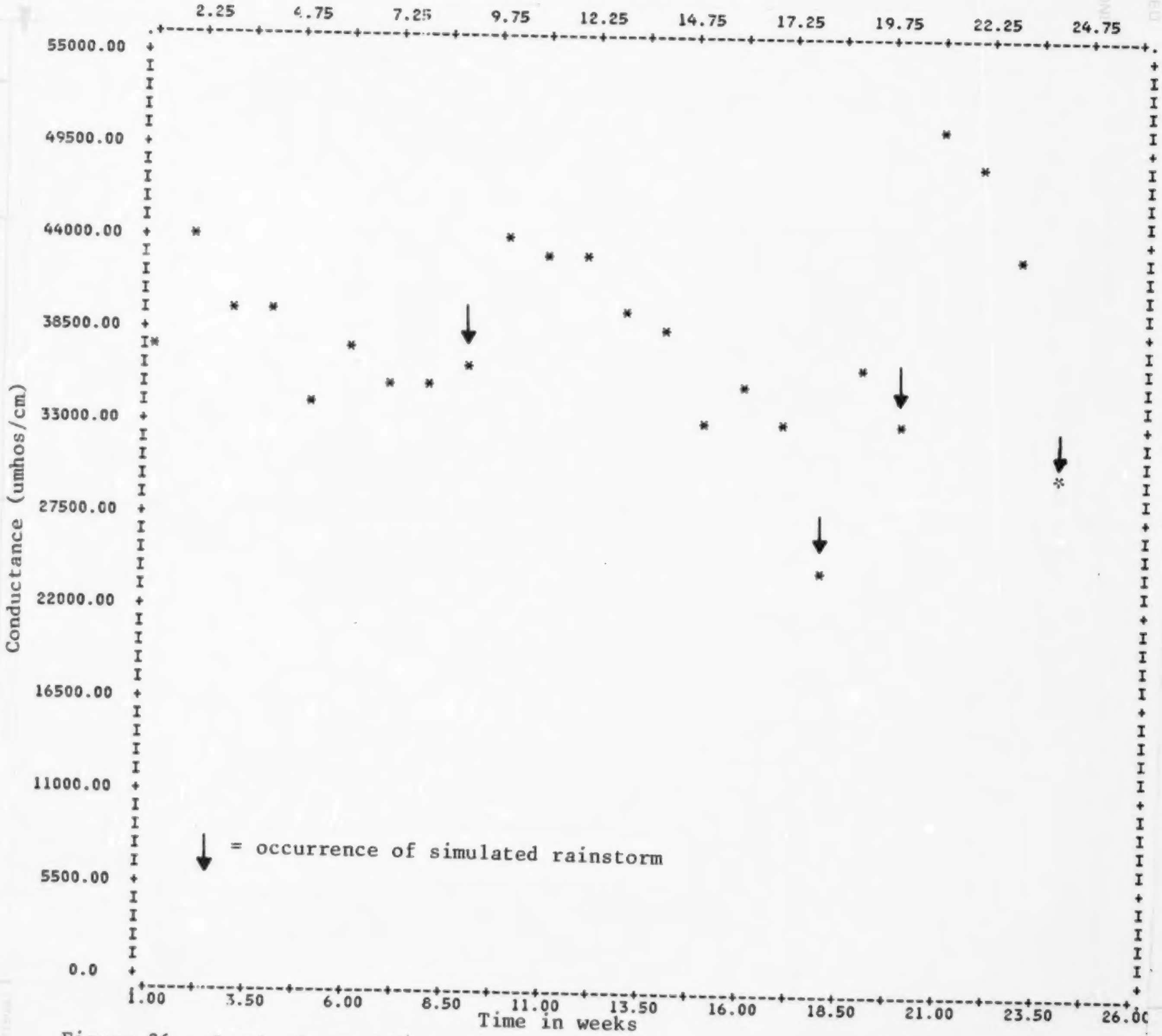

Figure 36. Graph of specific conductance as a function of time for Column 2 (mine waste control).

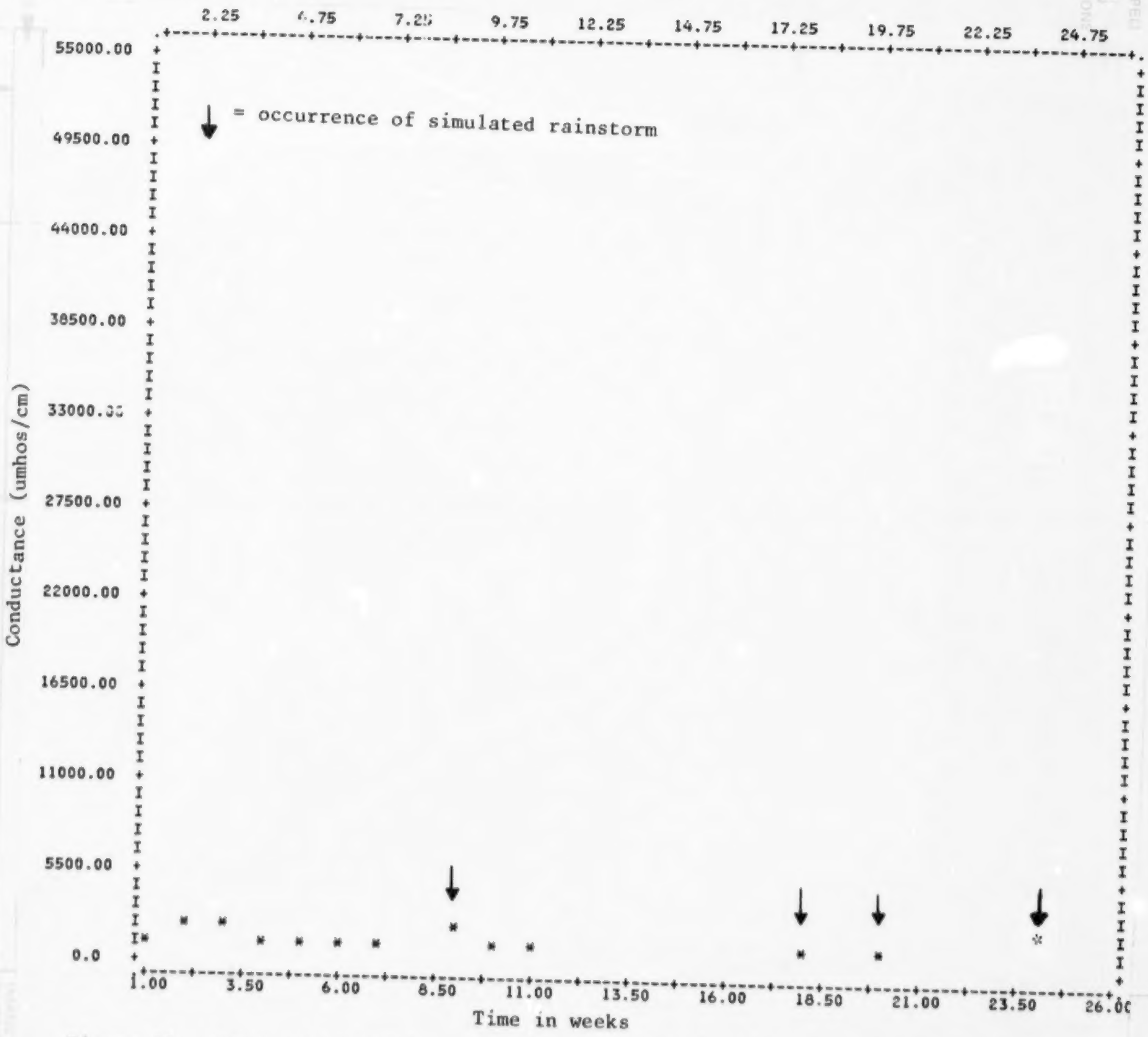

Figure 37. Graph of specific conductance as a function of time for Column 3 (cover soil control).

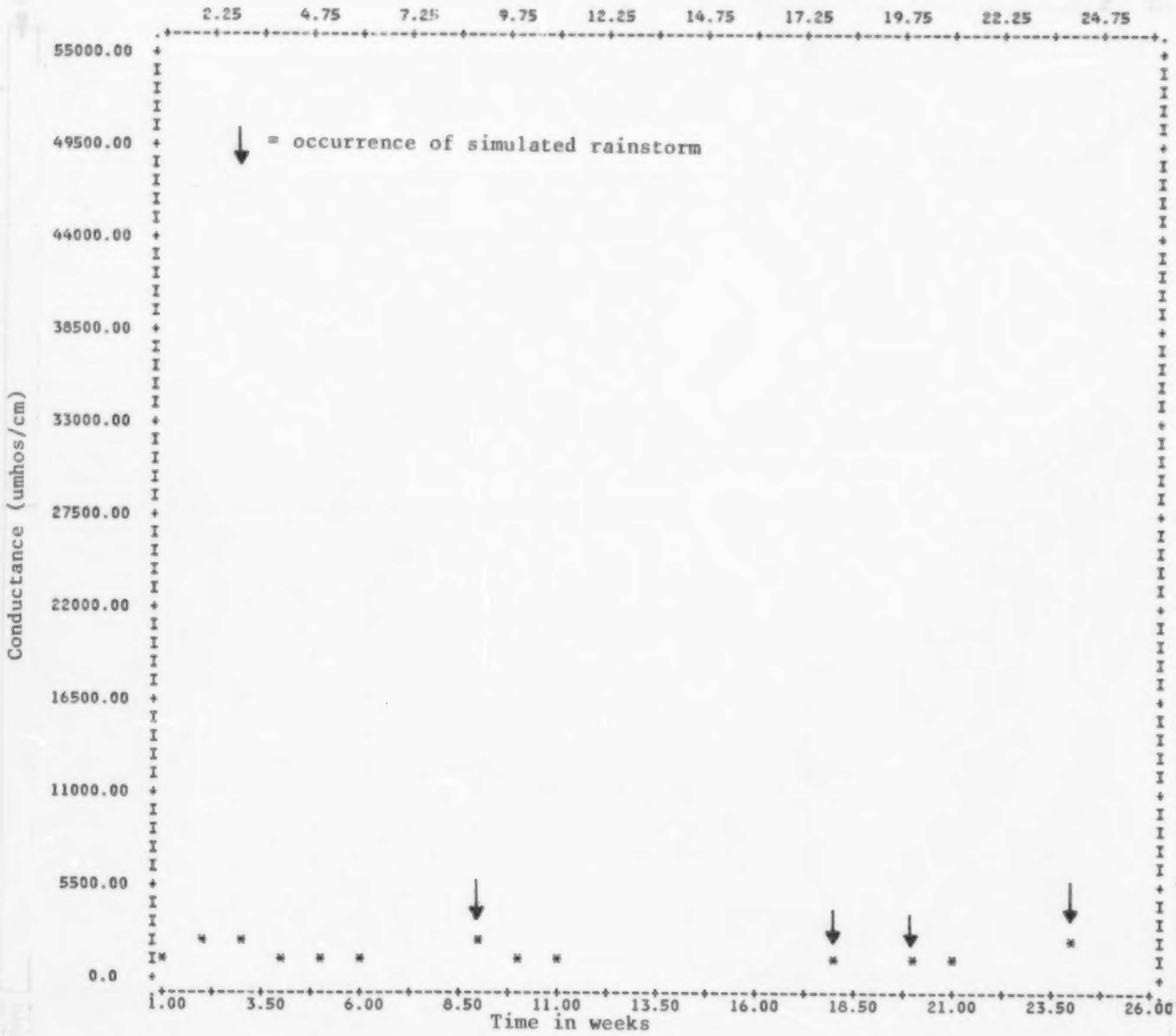

Figure 38. Graph of specific conductance as a function of time for Column 4 (cover soil control).

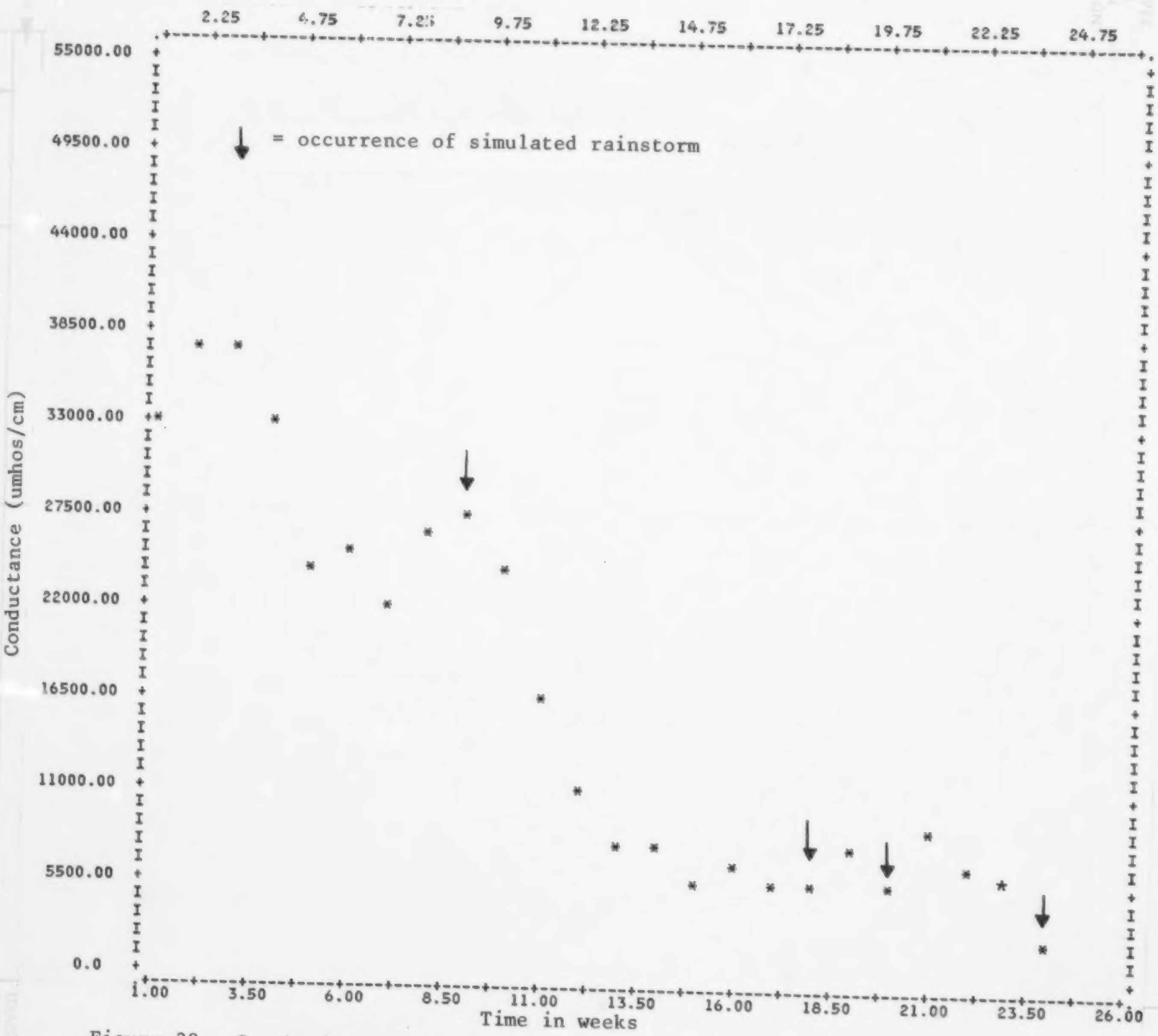

Figure 39. Graph of specific conductance as a function of time for Column 5 (incorporated amendments).

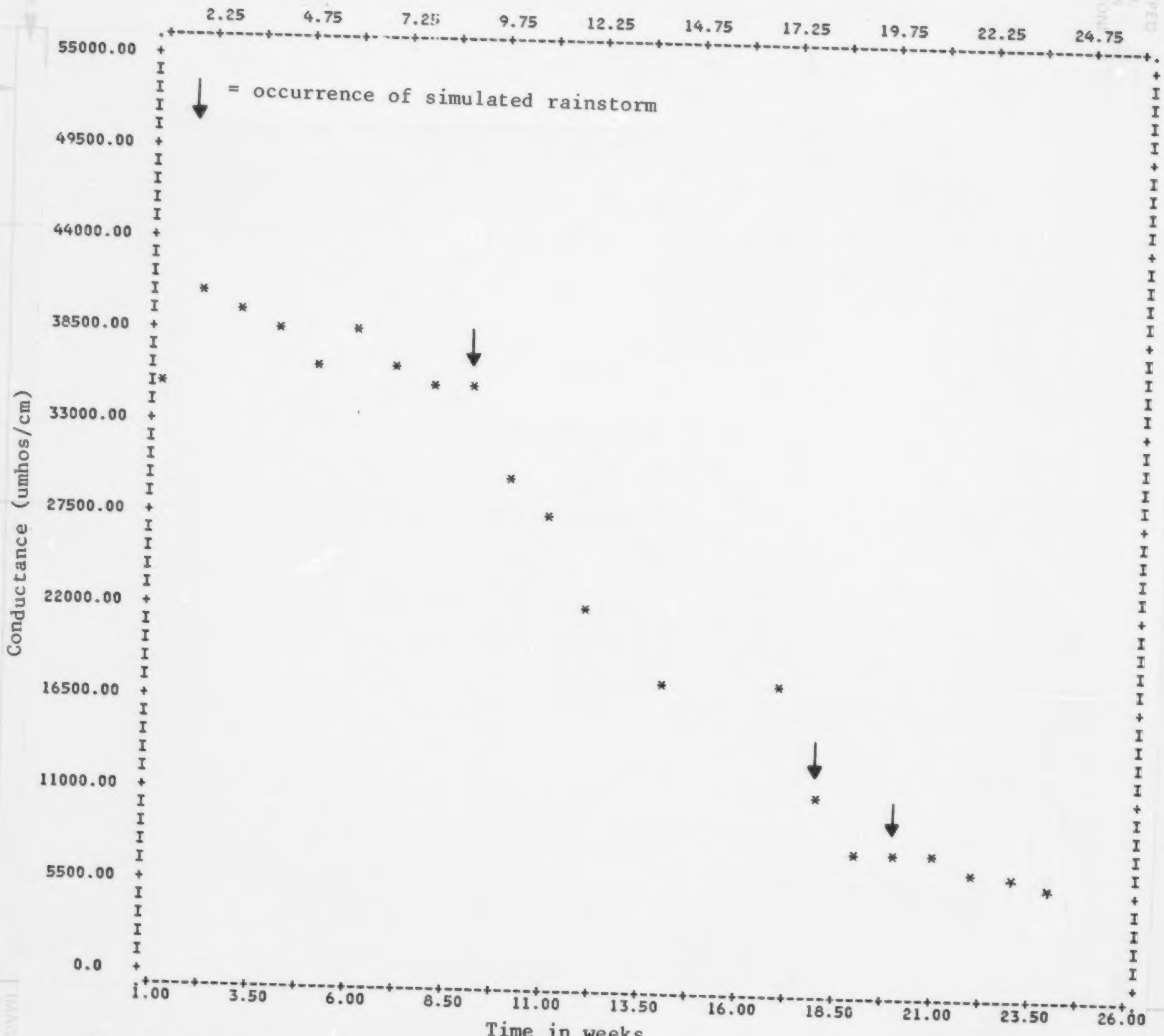

Figure 40. Graph of specific conductance as a function of time for Column 6 (incorporated amendments).

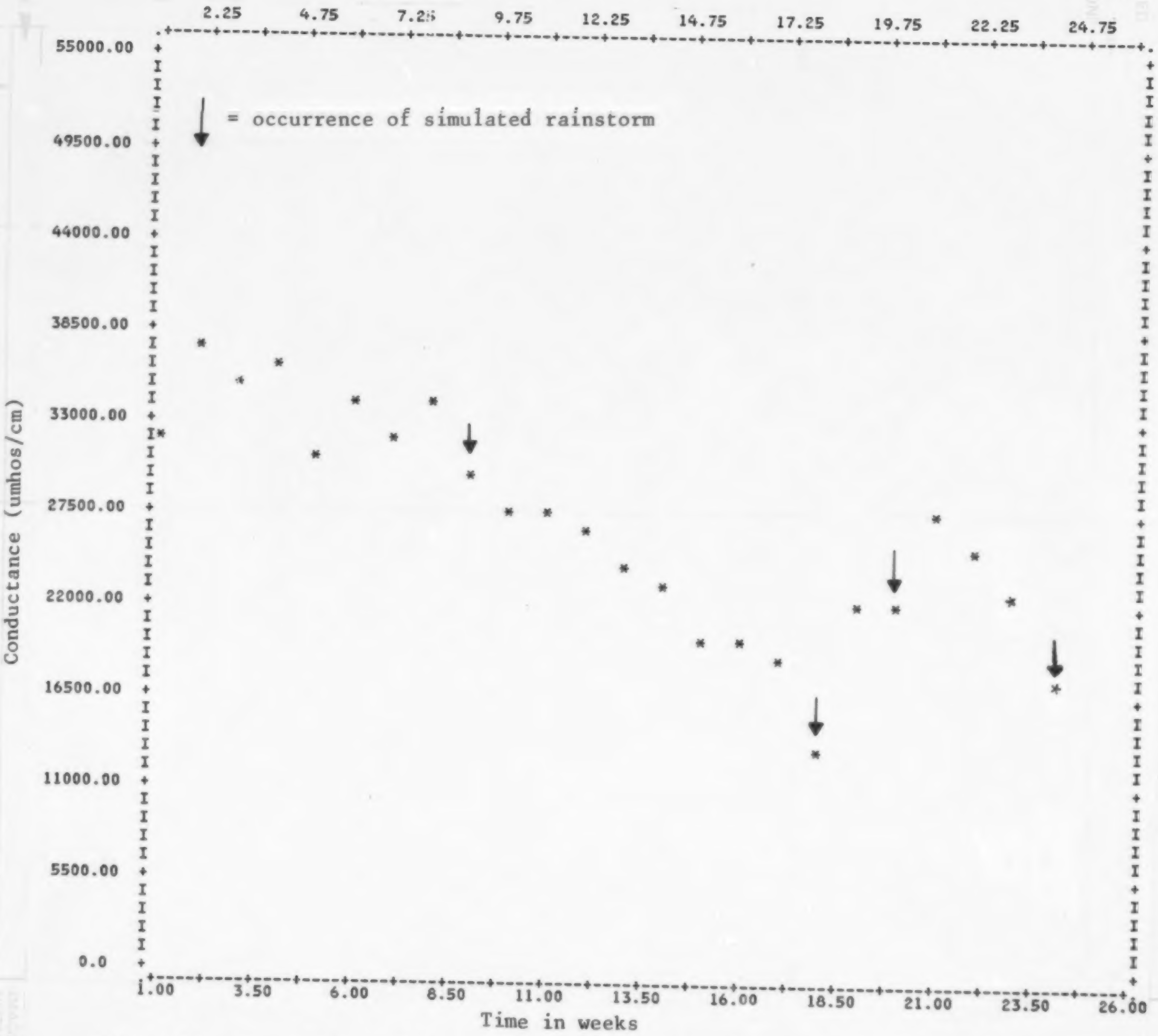

Figure 41. Graph of specific conductance as a function of time for Column 7 (layered amendments).

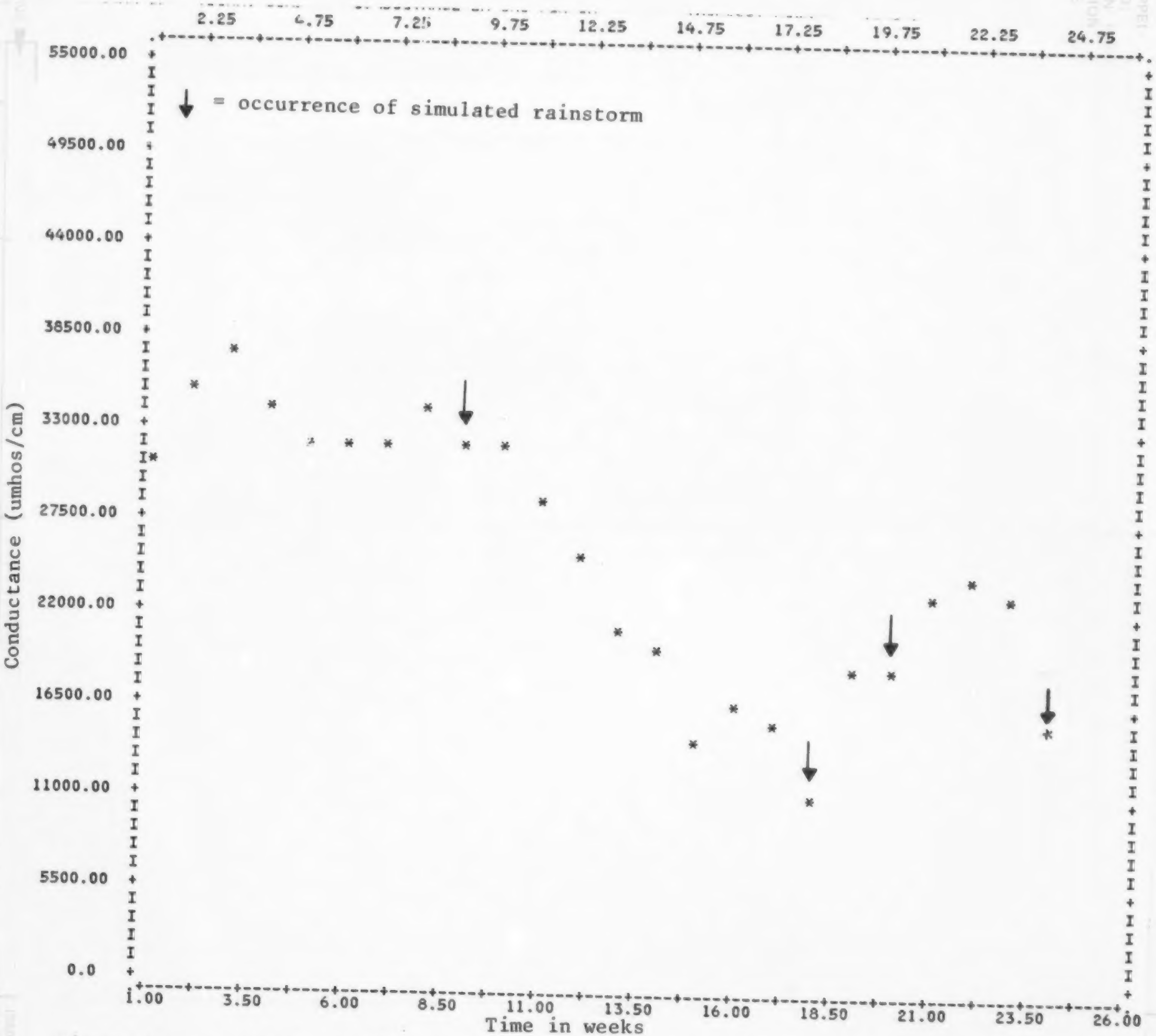

Figure 42. Graph of specific conductance as a function of time for Column 8 (layered amendments).

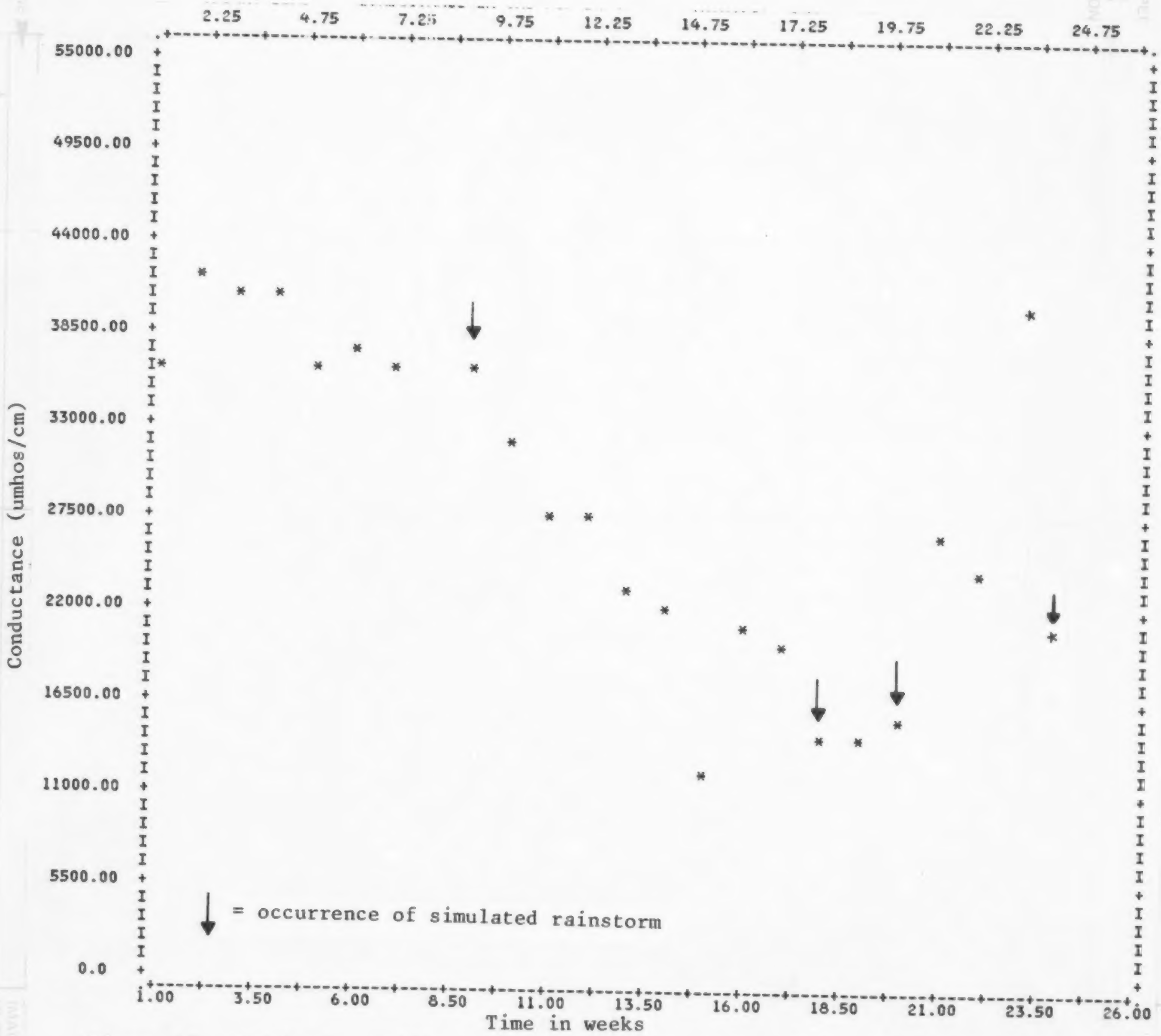

Figure 43. Graph of specific conductance as a function of time for Column 9 (0.46-m (1.5-ft) cover soil layer).

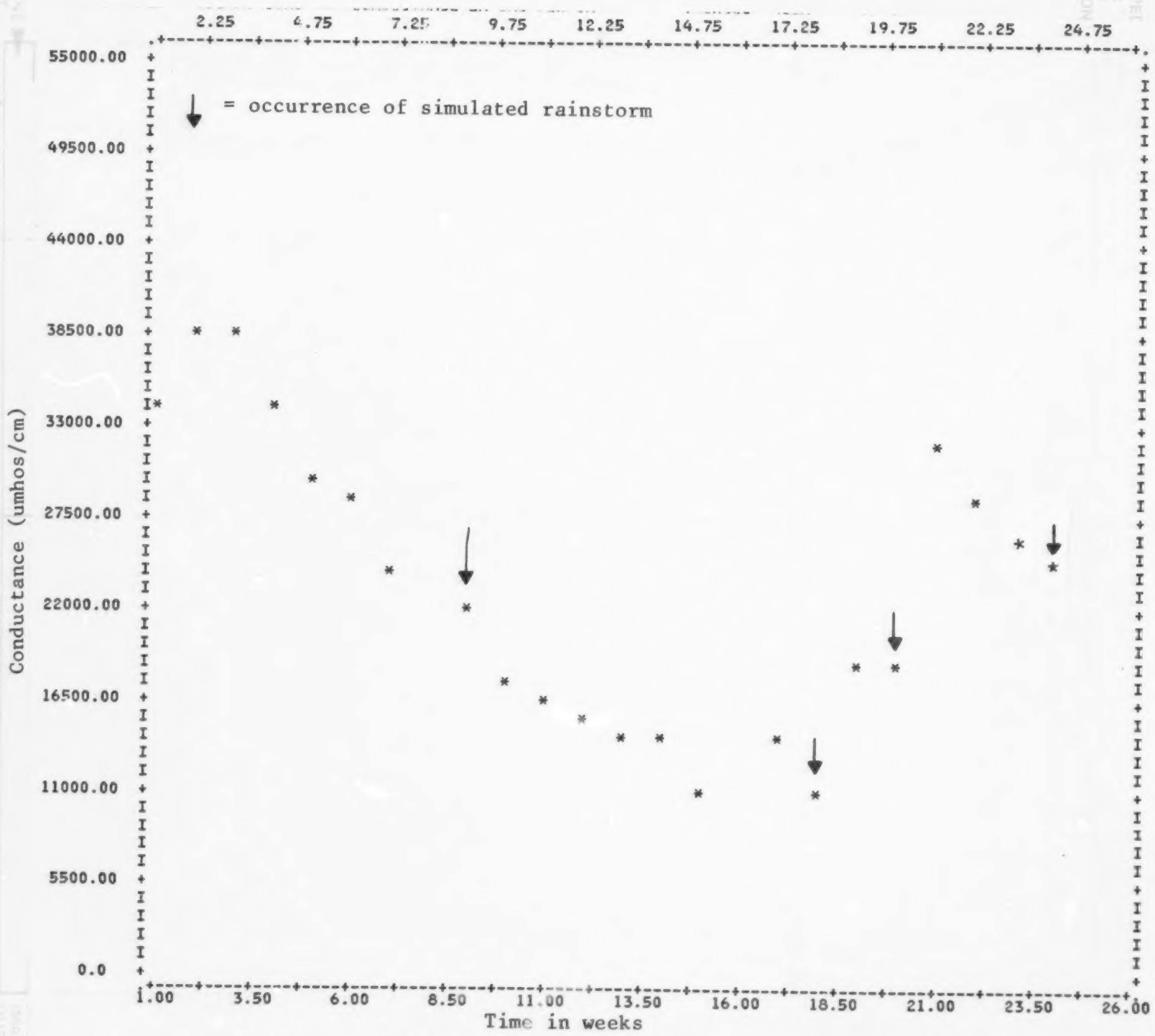

Figure 44. Graph of specific conductance as a function of time for Column 10 (0.46-m (1.5-ft) cover soil layer).

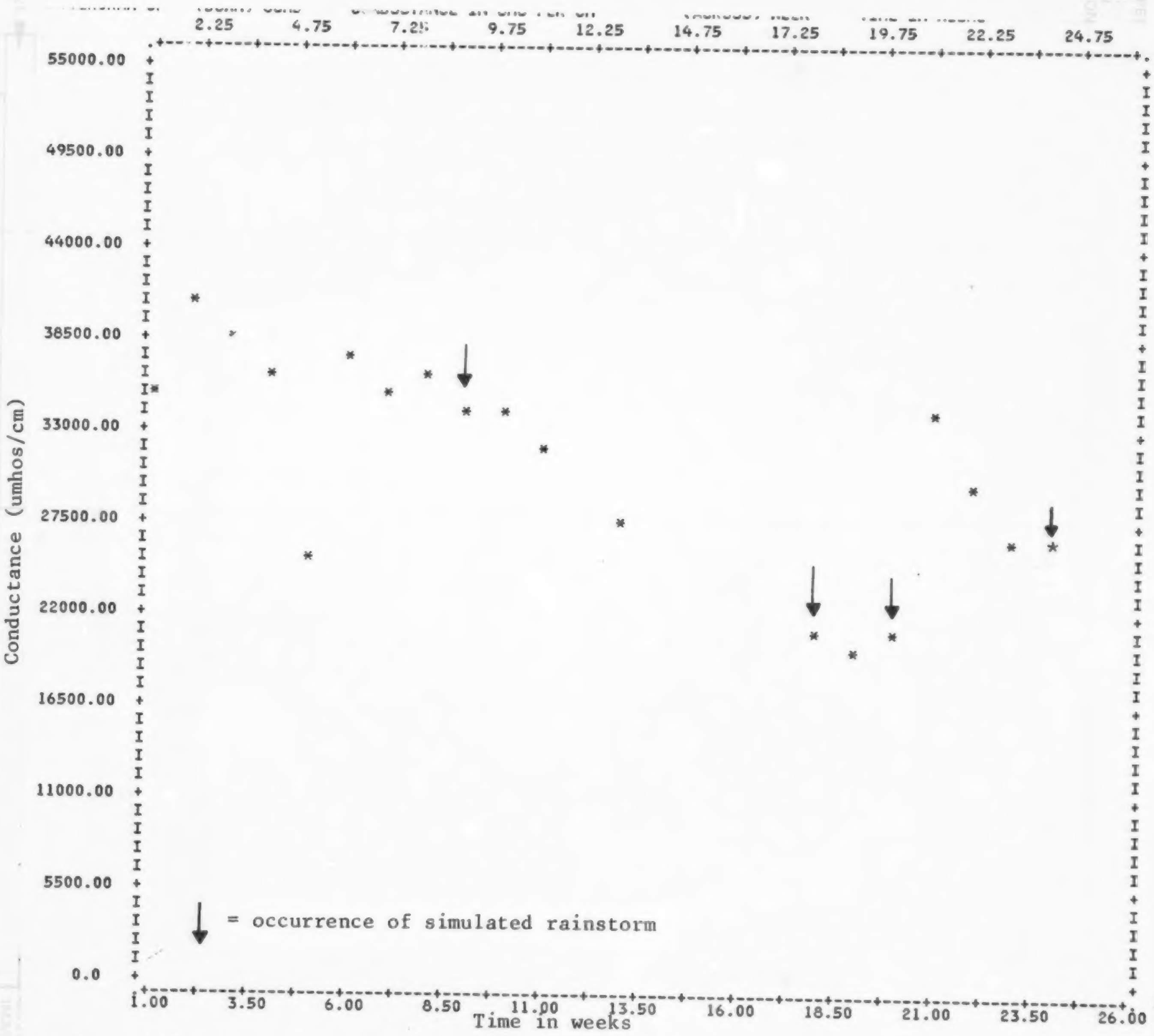

Figure 45. Graph of specific conductance as a function of time for Column 11 (0.61-m (2-ft) cover soil layer).

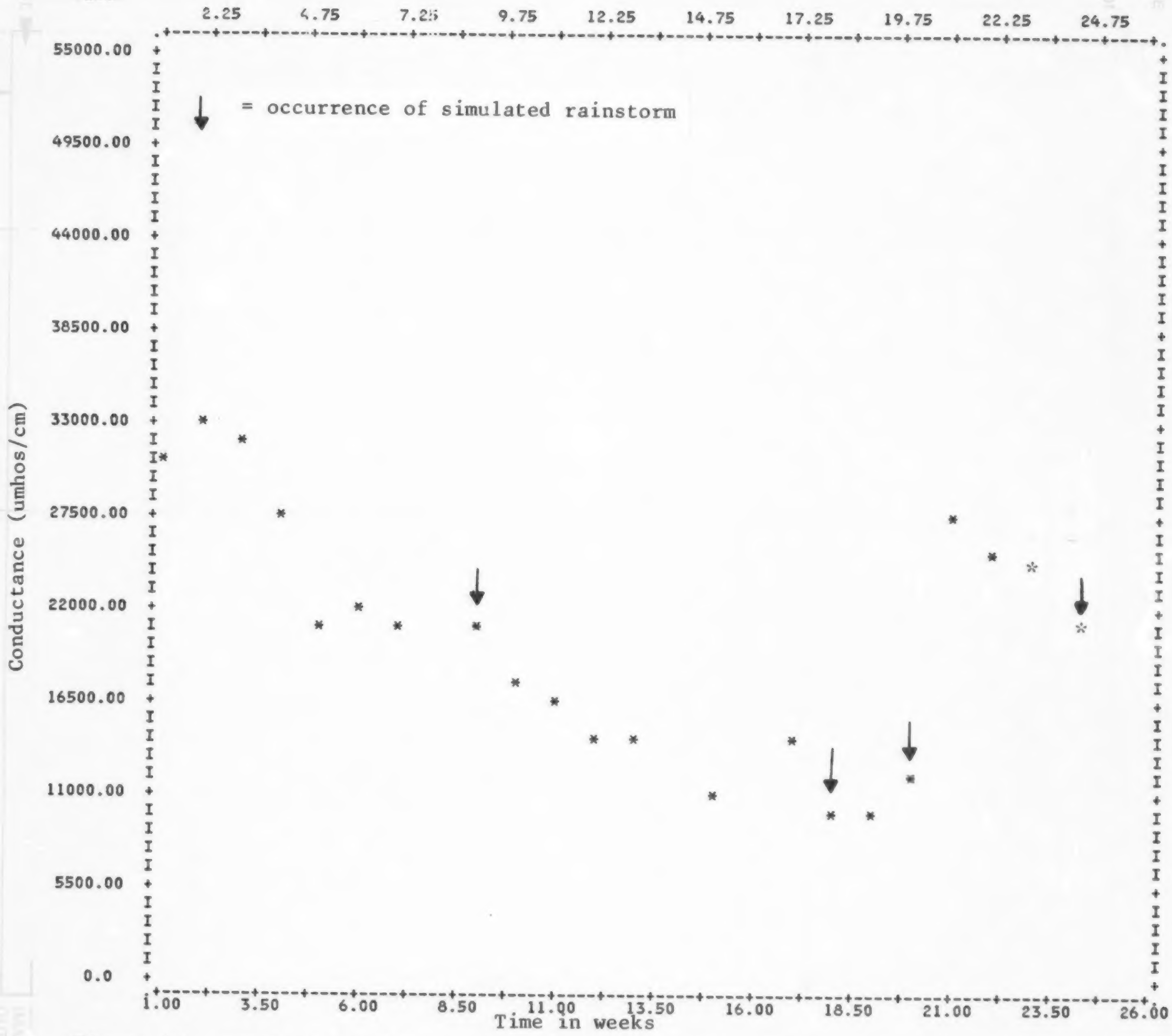

Figure 46. Graph of specific conductance as a function of time for Column 12 (0.61-m (2-ft) cover soil layer).

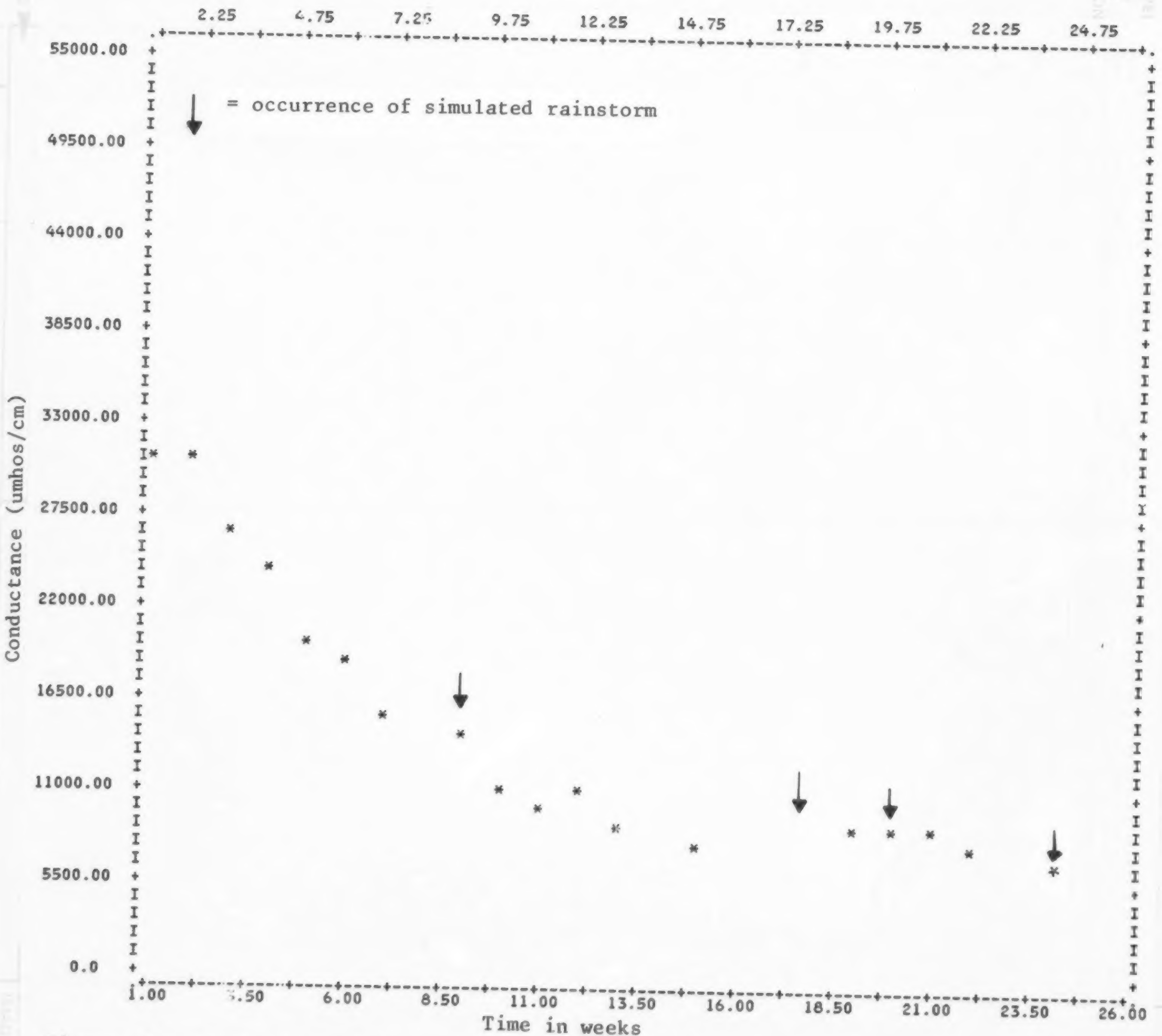

Figure 47. Graph of specific conductance as a function of time for Column 13 (1.2-m (4-ft) cover soil layer).

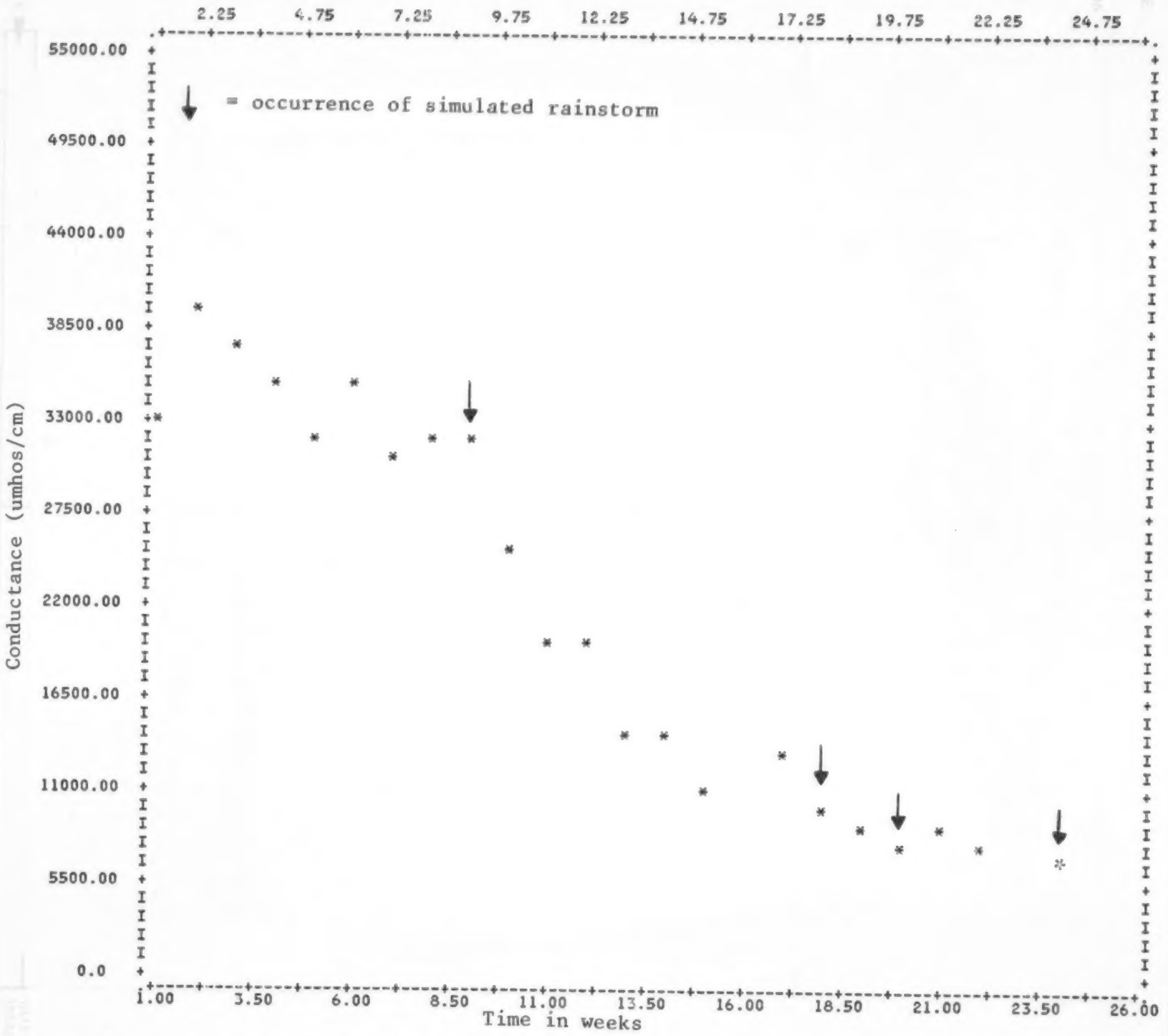

Figure 48. Graph of specific conductance as a function of time for Column 14 (1.2-m (4-ft) cover soil layer).

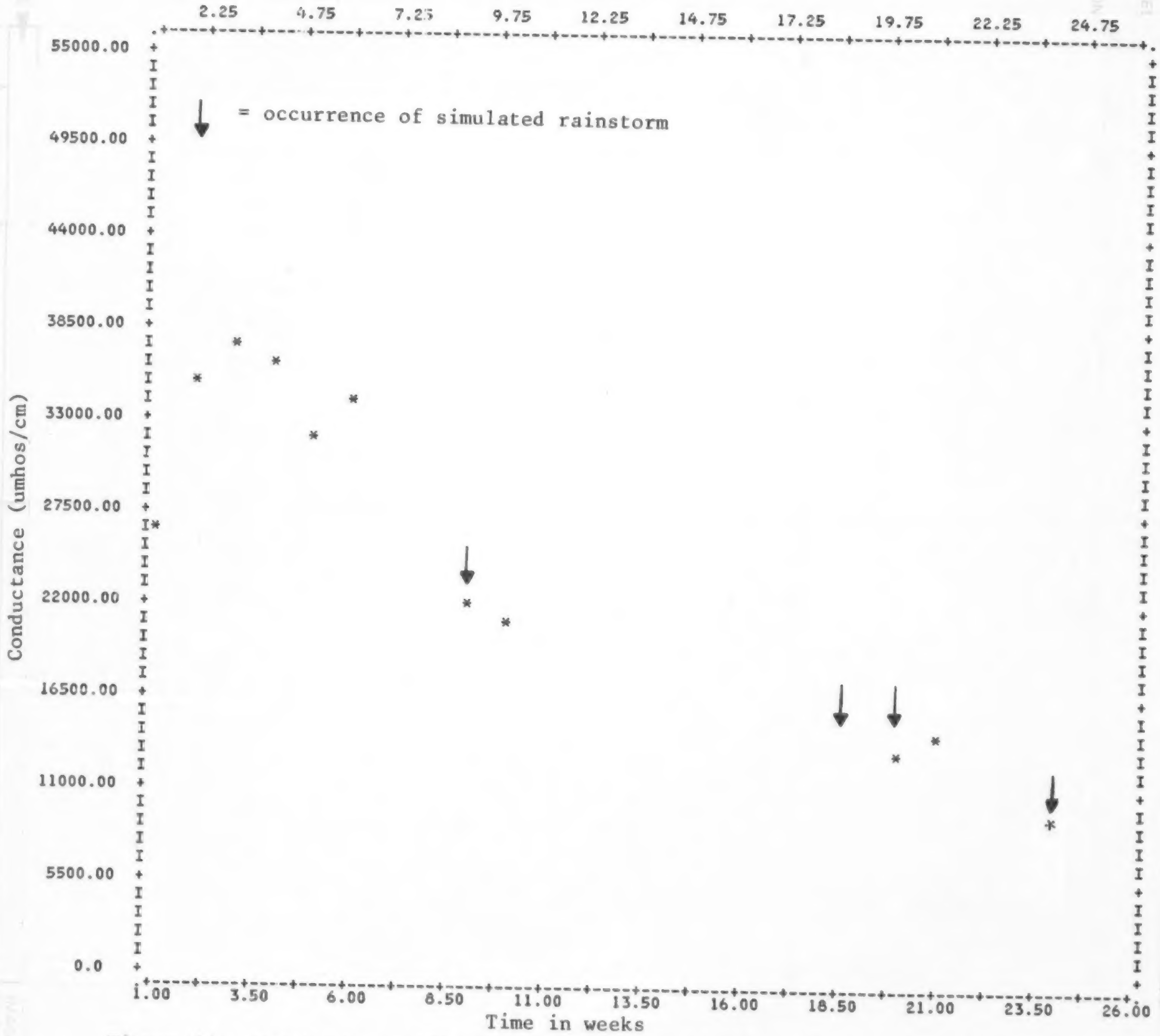

Figure 49. Graph of specific conductance as a function of time for Column 15 (2.5-cm (1-in) neutralizing layer).

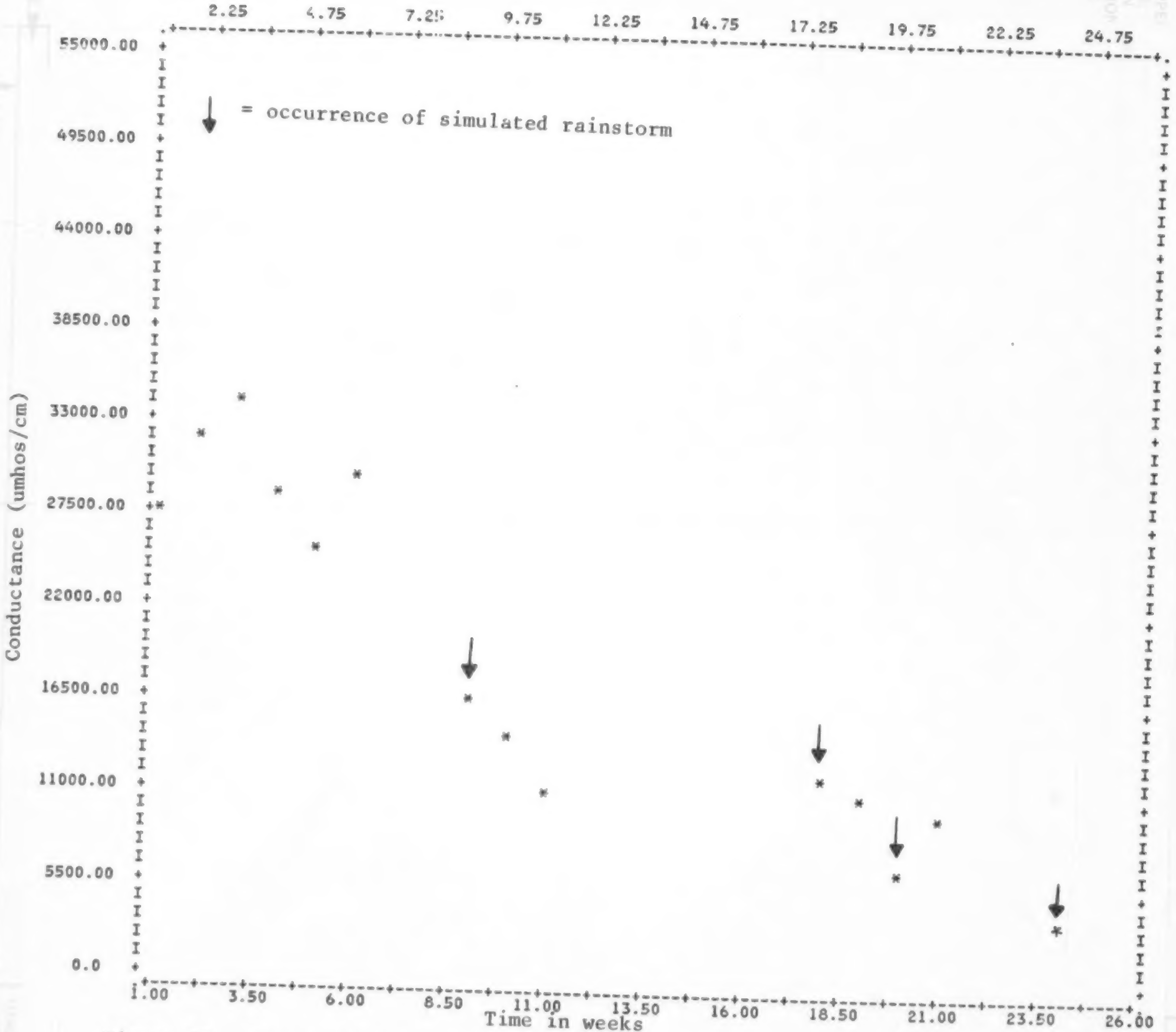

Figure 50. Graph of specific conductance as a function of time for Column 16 (cover soil/mine waste control).

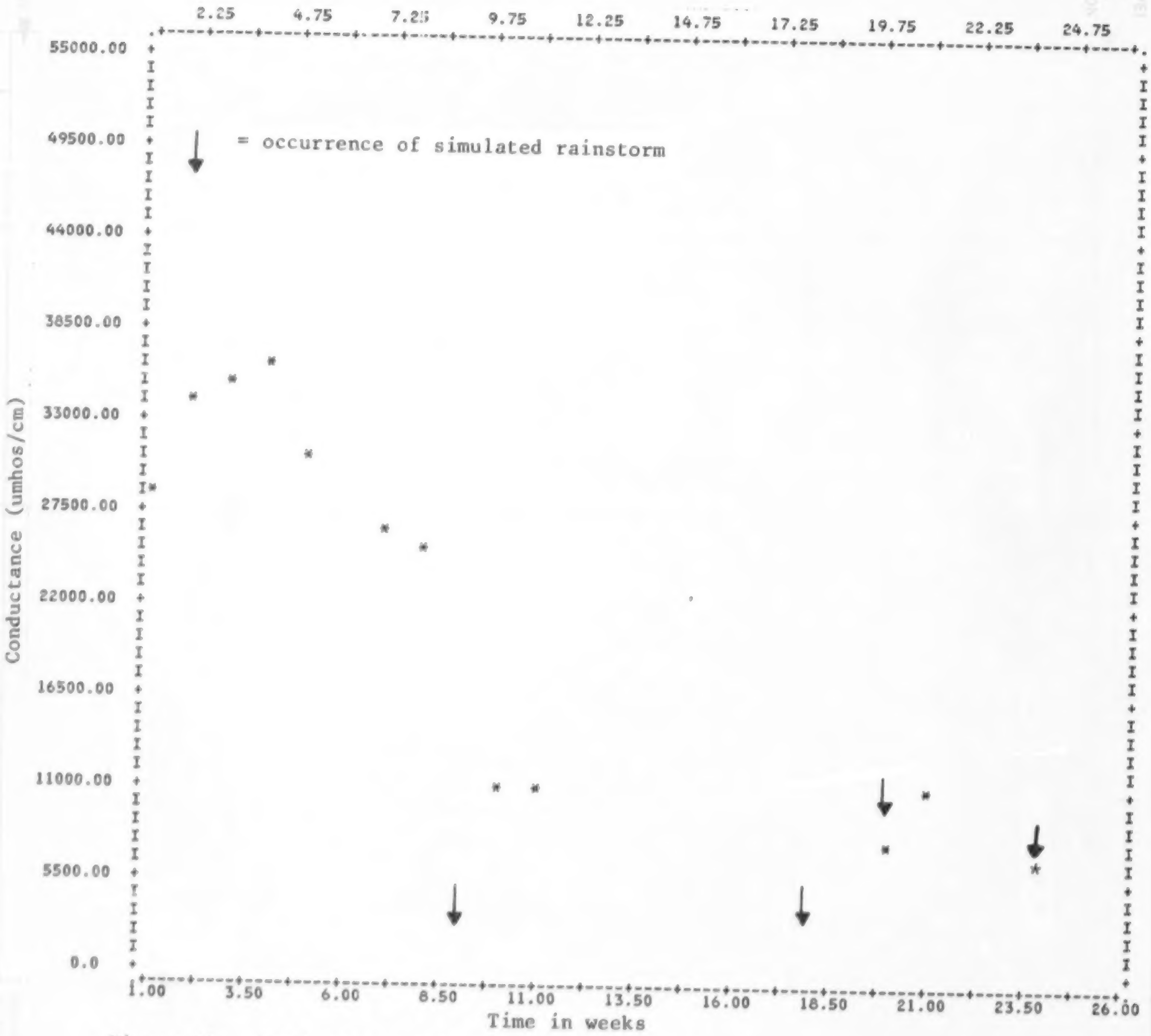

Figure 51. Graph of specific conductance as a function of time for Column 17 (5.0-cm (2-in) neutralizing layer).

the conductance of the leachate generated from the mine waste control columns (columns 1 and 2) ranged from a maximum of almost 55,000 umhos/cm to a minimum of 24,400 umhos/cm (Figures 35 and 36), while leachate from the soil control columns had a conductance averaging between 1,000 and 1,500 umhos/cm (Figures 37 and 38) which is considered to be characteristic of a normal top soil.

The specific conductance of the leachate from the experimental columns containing lime, sludge and fertilizer incorporated into the upper strata of the mine waste (columns 5 and 6) ranged between 33,000 and 40,000 umhos/cm at the beginning of the study and decreased steadily throughout the study period with a final conductance of less than 5,600 umhos/cm being noted (Figures 39 and 40). In comparison, the conductance of leachate generated from the columns containing the same amendments in a layered configuration (columns 7 and 8) ranged from an initial conductance of approximately 31,000 umhos/cm to a final conductance of approximately 16,500 umhos/cm (Figures 41 and 42). By comparing Figures 39 through 42, it can be visually noted that the rate of decrease in conductance with respect to time is considerably greater for the columns containing the amendments incorporated into the upper strata of the mine waste (columns 5 and 6) than for the columns containing these same amendments layered over the mine waste. This actual rate of decrease in leachate specific conductance for columns 5 and 6 was over two times as great as that seen in columns 7 and 8.

Placement of a cover soil layer over the mine waste also reduced leachate conductance. The most significant reduction observed when a layer of cover soil was used was in leachate data from columns containing 1.2-m (4-ft) layer of cover soil placed over the mine waste (columns 13 and 14). Conductance in the leachate from these columns ranged from a maximum of approximately 39,000 umhos/cm to a minimum of 6,830 umhos/cm recorded at the end of the study period. A steady decrease in the conductance of the leachate from columns 13 and 14 (Figures 47 and 48) can be seen in contrast to the data from columns 9 through 12 where thinner cover soil layers were used (Figures 43 through 46). The conductance of the leachate resulting from columns containing 0.61-m (2 ft) of cover soil over the mine waste (columns 11 and 12) ranged from slightly above 40,000 umhos/cm to approximately 11,000 umhos/cm (Figures 45 and 46). This compared very closely to the range of conductance data for leachate from columns 9 and 10 containing 0.5 m (1.5 ft) cover soil layer. The most significant rate of decrease in leachate conductance was realized when 1.2 m (4 ft) layer of cover soil was used. This configuration produced a leachate with a conductance from 1.5 to 2.5 times lower than that generated when thinner layers of cover soil were used (columns 9 through 12).

The conductance of the leachates generated from columns containing a neutralizing layer positioned between the mine waste and cover soil material (columns 15 and 17) showed little variation over the study period. A general decrease was noted, but differentiation between the experimental and control columns was minor. The conductance of the leachate from column 15, containing a 2.5-cm (1-in) lime layer, ranged from 36,900 to 10,100 umhos/cm (Figure 49). When a 5-cm (2-in) layer of lime was used (column 17), leachate conductance ranged from 36,200 to 5,680 umhos/cm (Figure 51). These ranges compared to that of 34,600 to 4,480 umhos/cm when no lime layer was used. The rate of decrease in leachate conductance over the study period for column 15, containing the 2.5-cm (1-in) neutralizing layer, was approximately 0.9 times greater than that of the control (column 16) while the rate of decrease for column 17, containing the 5-cm (2-in) neutralizing layer, was approximately 1.1 times greater than that of the control.

Metal Data

Initial metal analyses were performed on the mine waste and cover soil materials. These analyses involved combination acid extractions of each material, with concentrations determined by ICAP. The resulting data are recorded in Table 8. Based on these findings, the mean iron, copper, zinc and lead leachate loads were calculated for each pair of columns and compared to the control columns to determine the effects of the various treatment techniques in metal abatement.

TABLE 8. RESULTS OF METAL ANALYSIS OF THE MINE WASTE AND THE COVER SOIL MATERIAL

Variable	Mine Waste Dry Wt (mg/g)	Cover Soil Dry wt (mg/g)
Iron	200	40
Lead	4	1
Copper	10	1
Zinc	2	1

Iron--

Plots illustrating mean iron loads as a function of time for each treatment technique are shown in Figures 52 through 61. Mean iron loads of the leachate from the untreated pyrite mine waste control columns (columns 1 and 2) averaged 55 g/day and ranged from a maximum of approximately 200 g/day to a minimum of approximately 20 g/day (Figure 52). As noted with other variables, maximum iron loads corresponded to simulated 10-year, 24-hour rainstorm events. Iron loads in the leachate from the control columns containing only the cover soil material (columns 3 and 4) were consistently low and comparable to those of a normal topsoil.

In comparison to the mine waste control columns, the incorporation of lime, sludge and fertilizer into the top 15.2 cm (6 in) of the mine waste resulted in a decrease of approximately 67 percent in the mean iron load of the leachate. A maximum mean iron loading of 82 g/day was observed in the leachate from these experimental columns after the first simulated storm event with a minimum of 0.94 g/day noted seven weeks later (Figure 54). An average iron loading of 18 g/day was noted throughout the study period. In

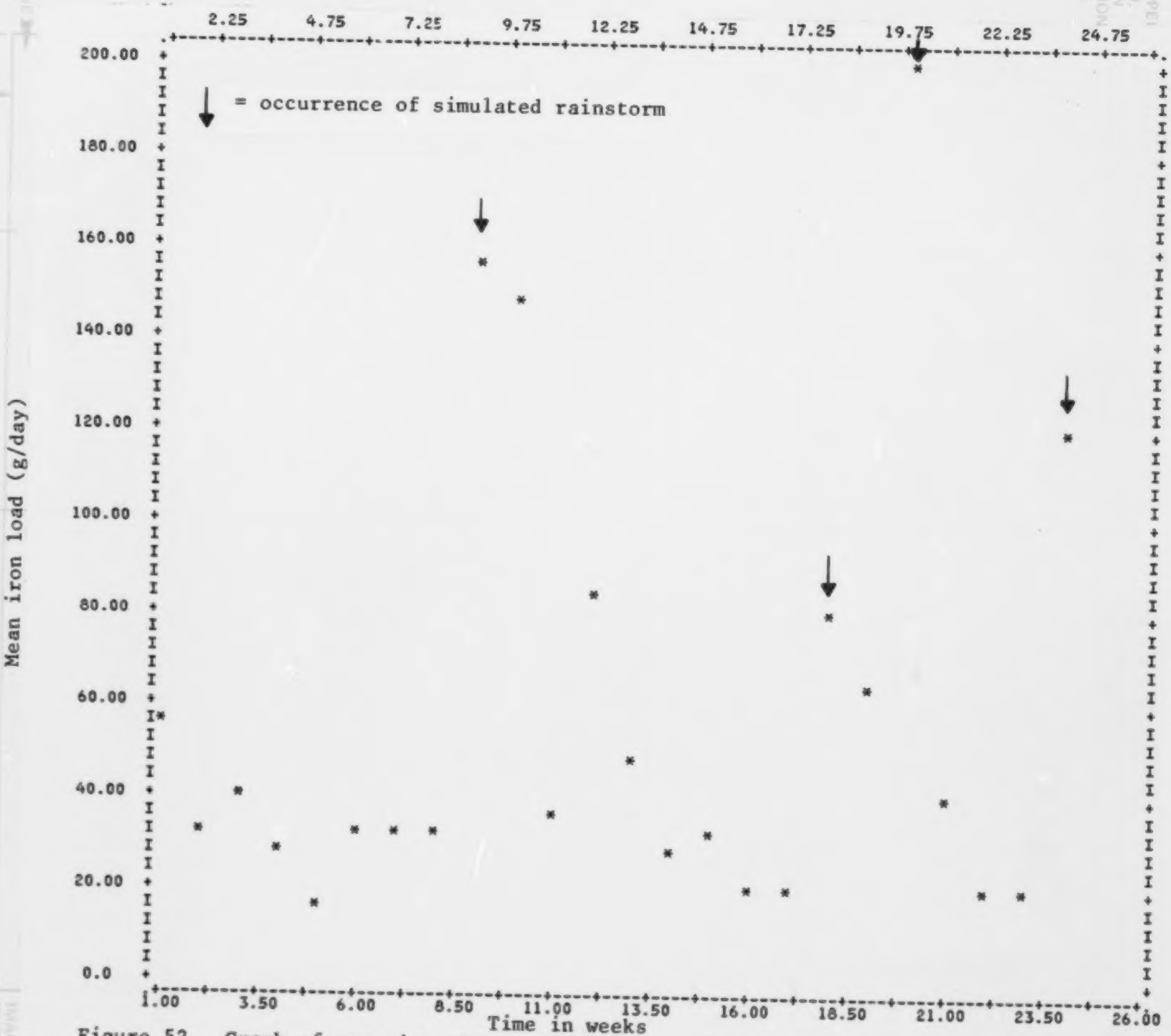

Figure 52. Graph of mean iron load as a function of time for Columns 1 and 2 (mine waste controls).

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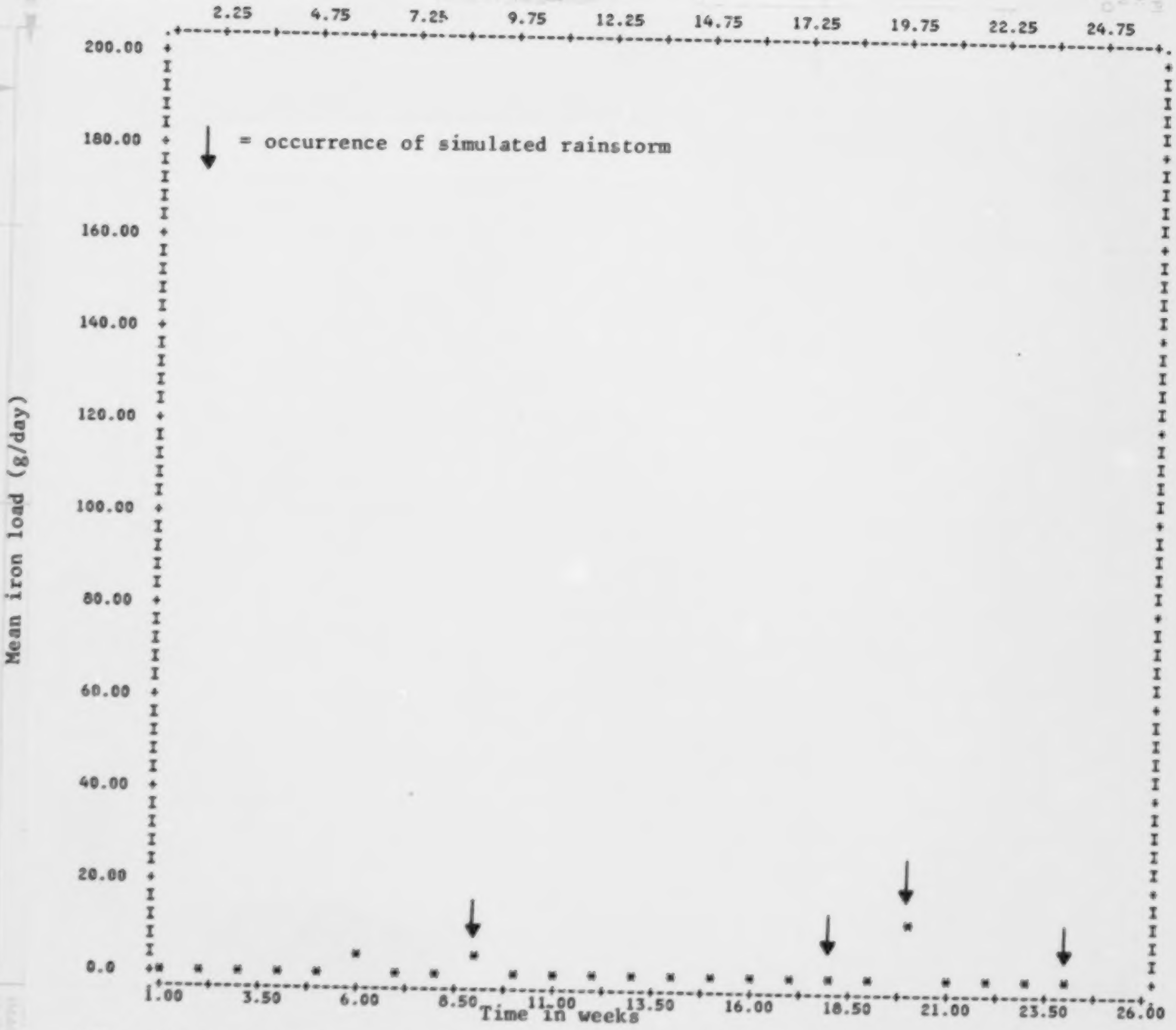

Figure 53. Graph of mean iron load as a function of time for Columns 3 and 4 (cover soil controls).

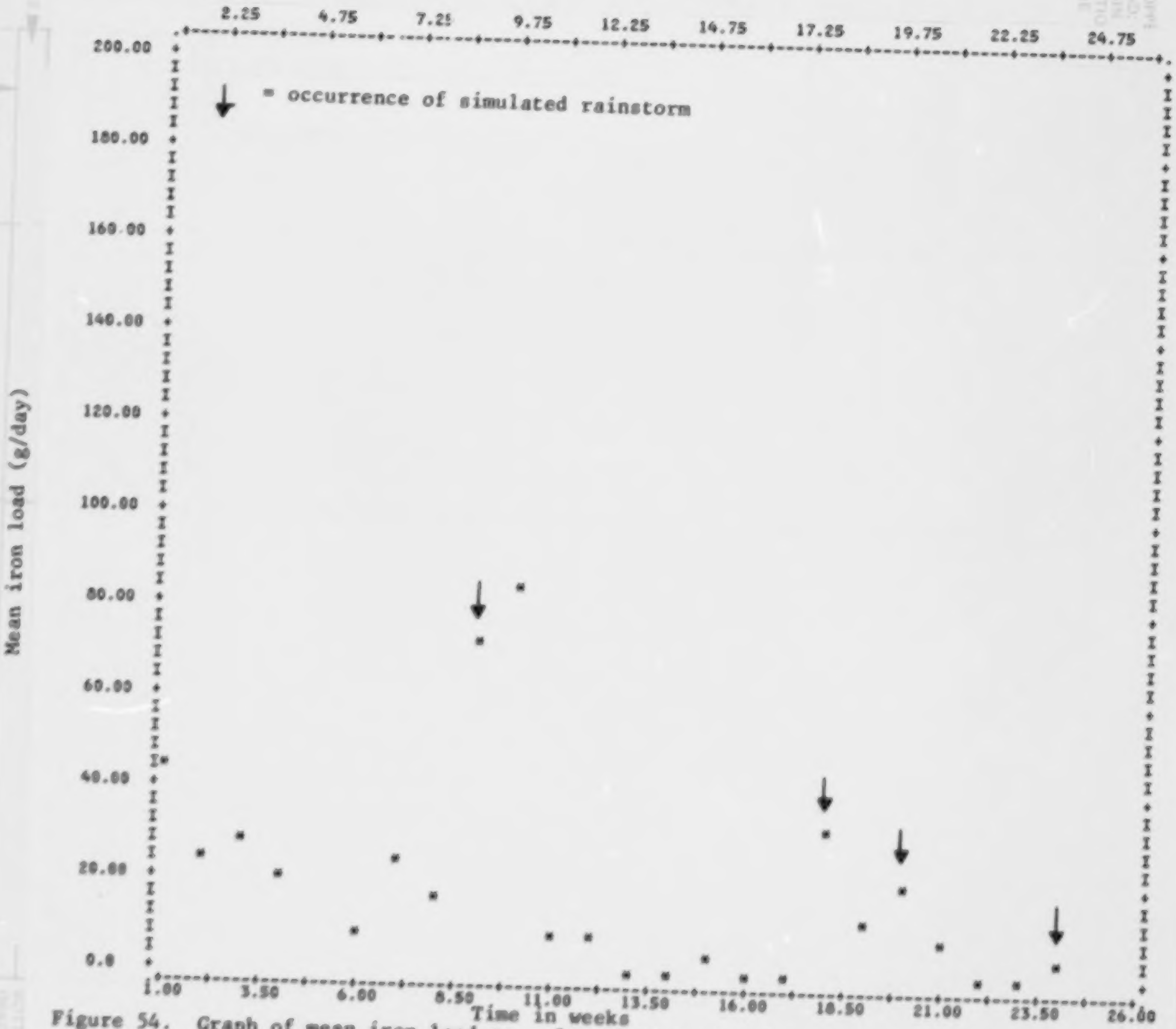

Figure 54. Graph of mean iron load as a function of time for Columns 5 and 6 (incorporated amendments).

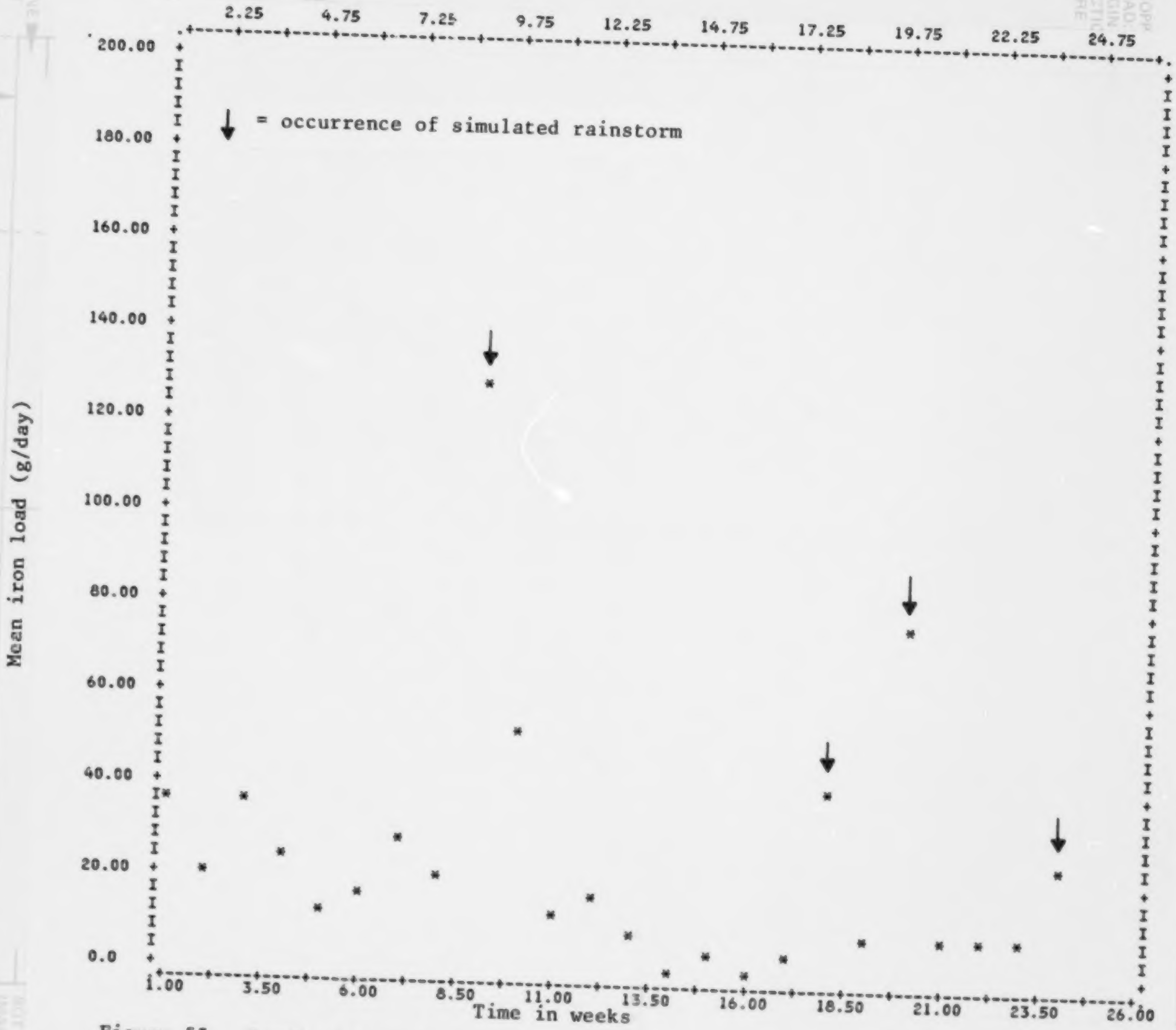

Figure 55. Graph of mean iron load as a function of time for Columns 7 and 8 (layered amendments).

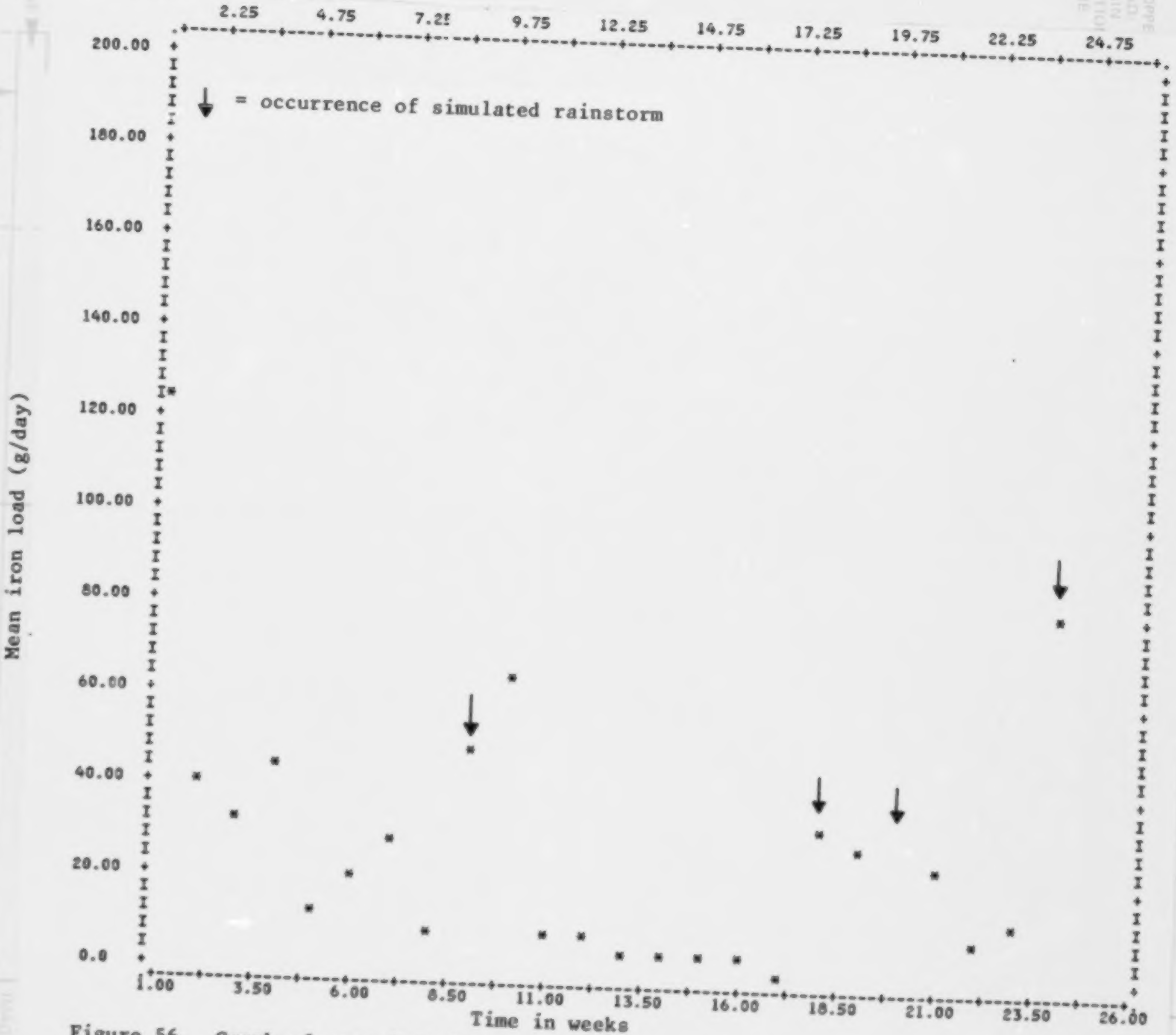

Figure 56. Graph of mean iron load as a function of time for Columns 9 and 10 (0.46-m (1.5-ft) cover soil layer).

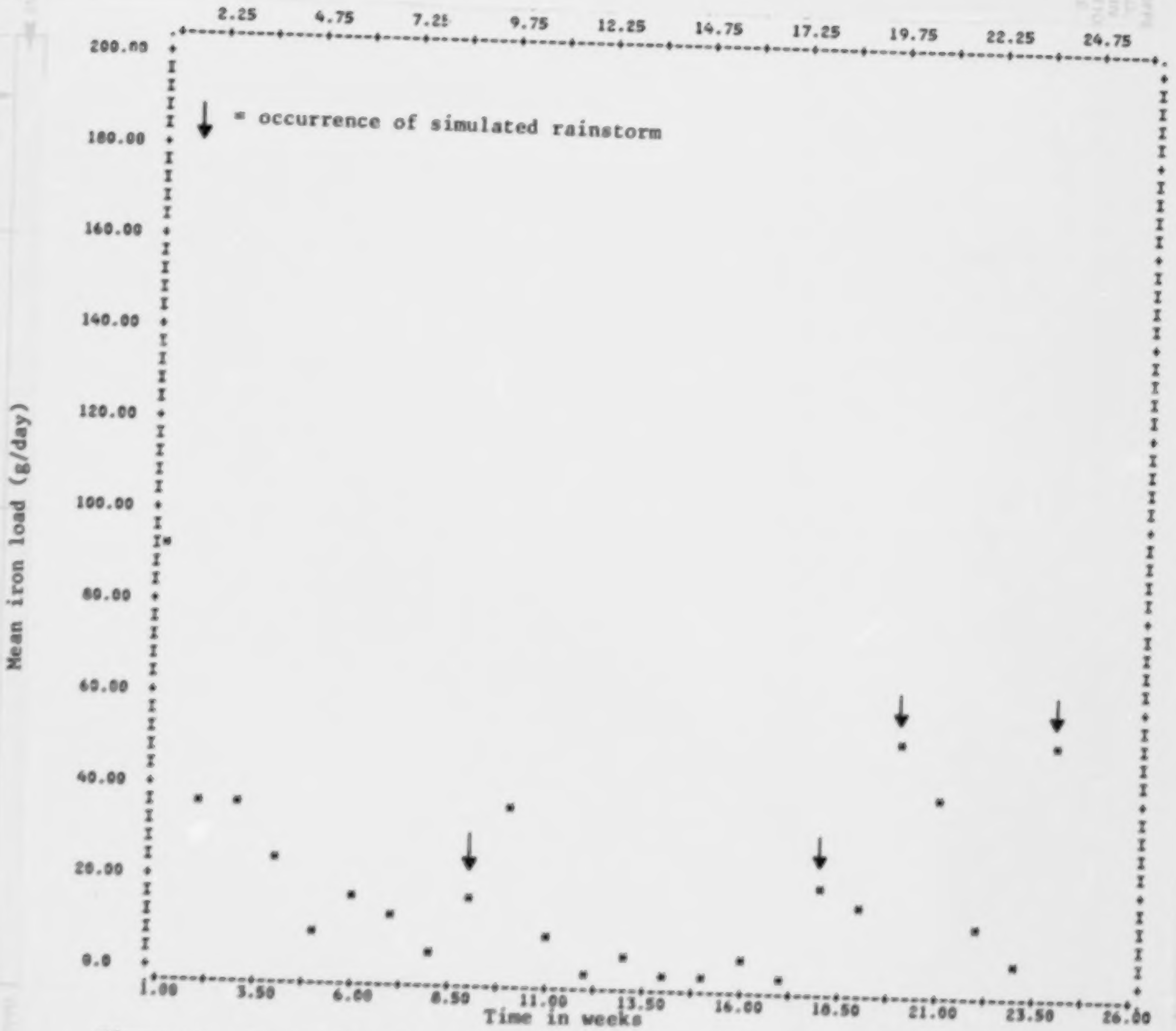

Figure 57. Graph of mean iron load as a function of time for Columns 11 and 12 (0.61-m (2-ft) cover soil layer).

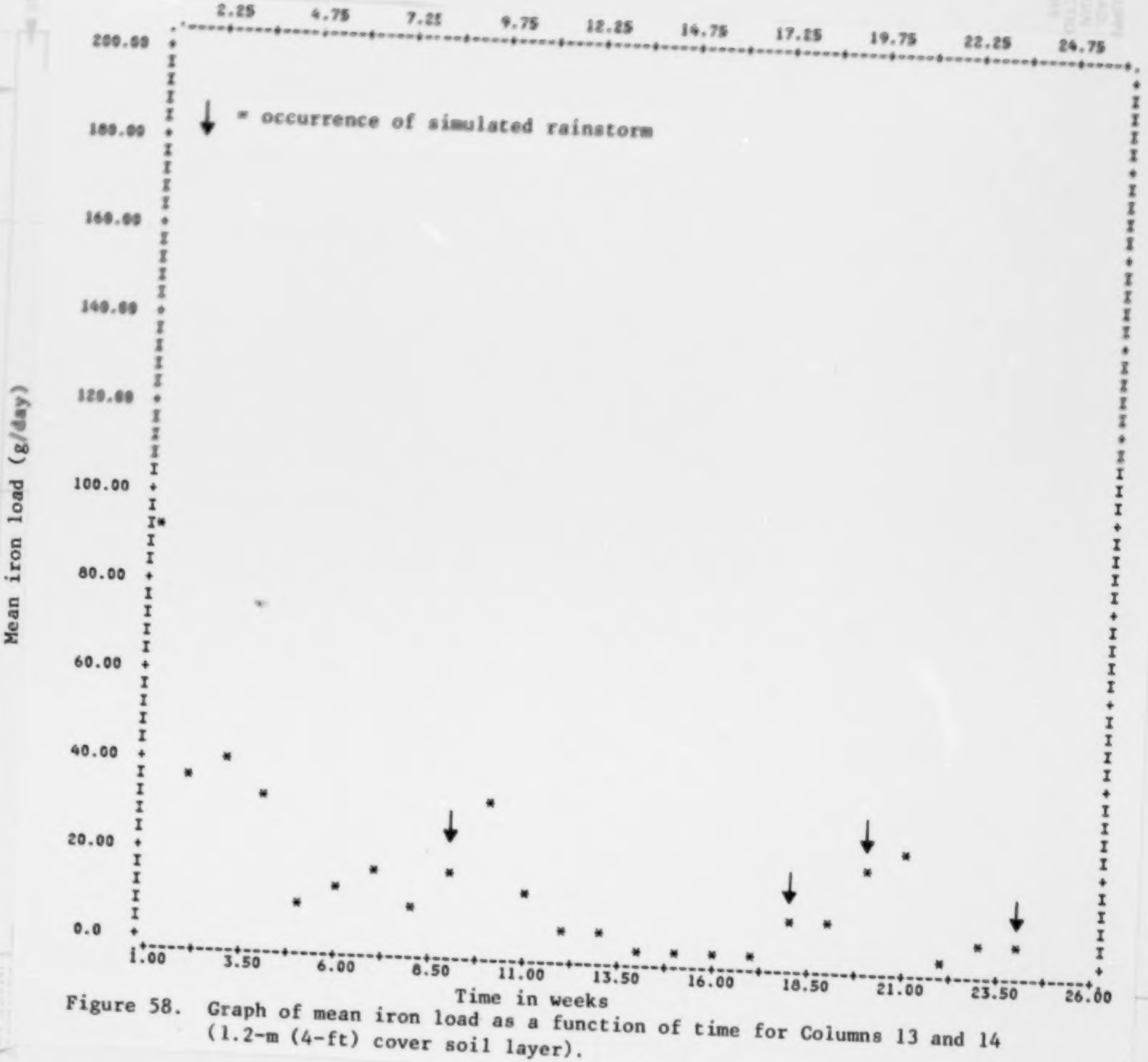

Figure 58. Graph of mean iron load as a function of time for Columns 13 and 14 (1.2-m (4-ft) cover soil layer).

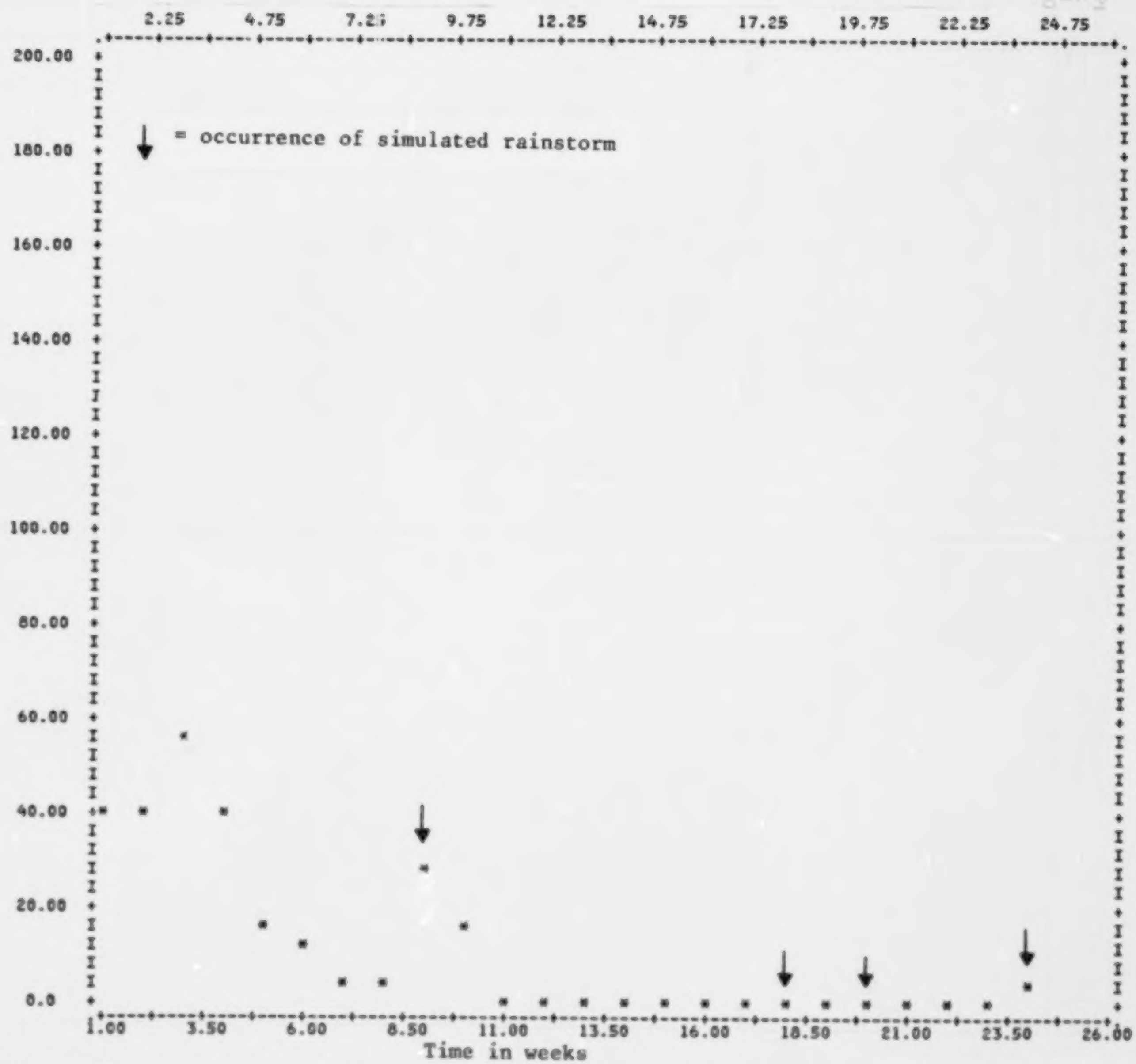

Figure 59. Graph of iron load as a function of time for Column 15 (2.5-cm (1-in) neutralizing layer).

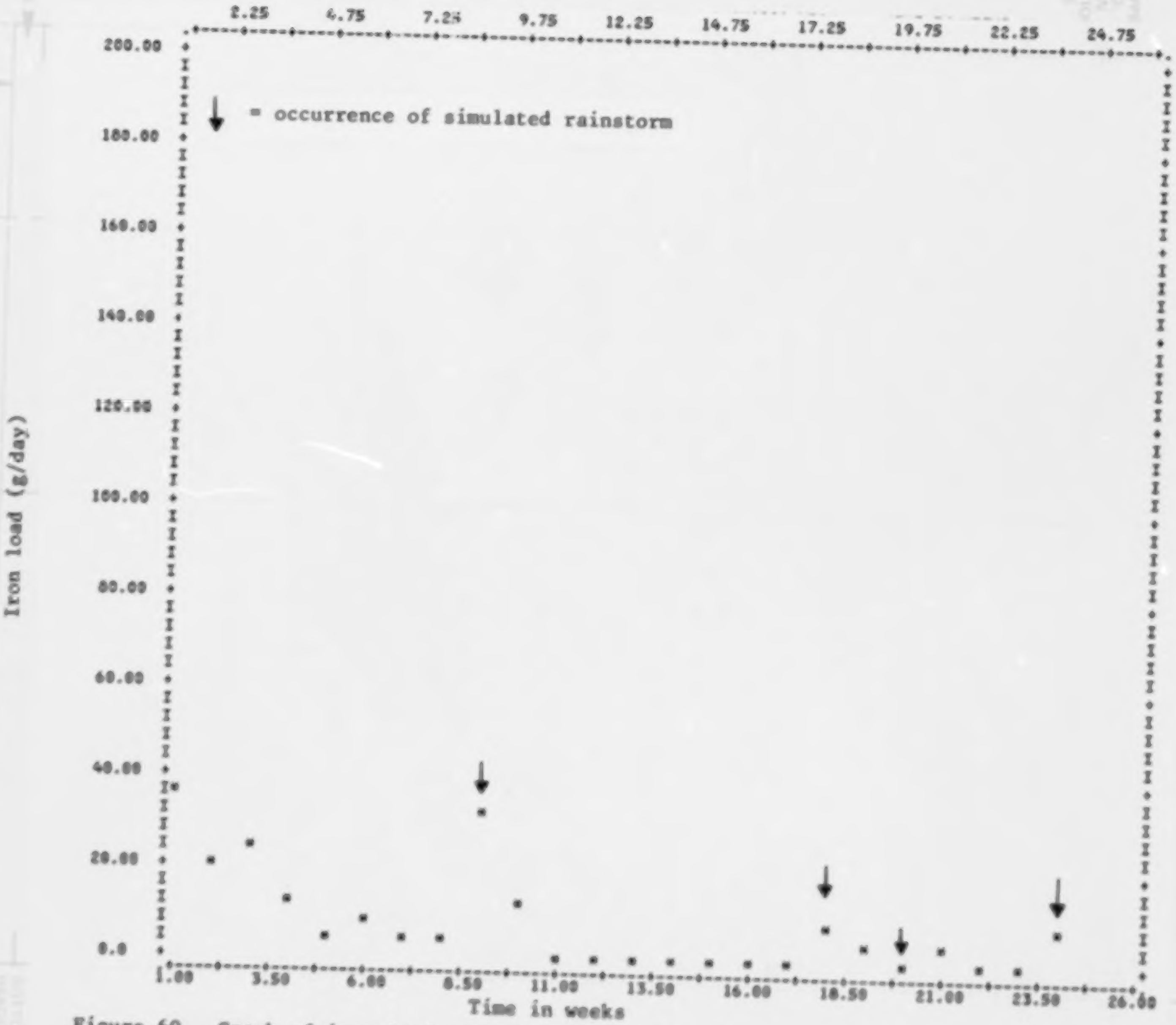

Figure 60. Graph of iron load as a function of time for Column 16 (cover soil/mine waste control).

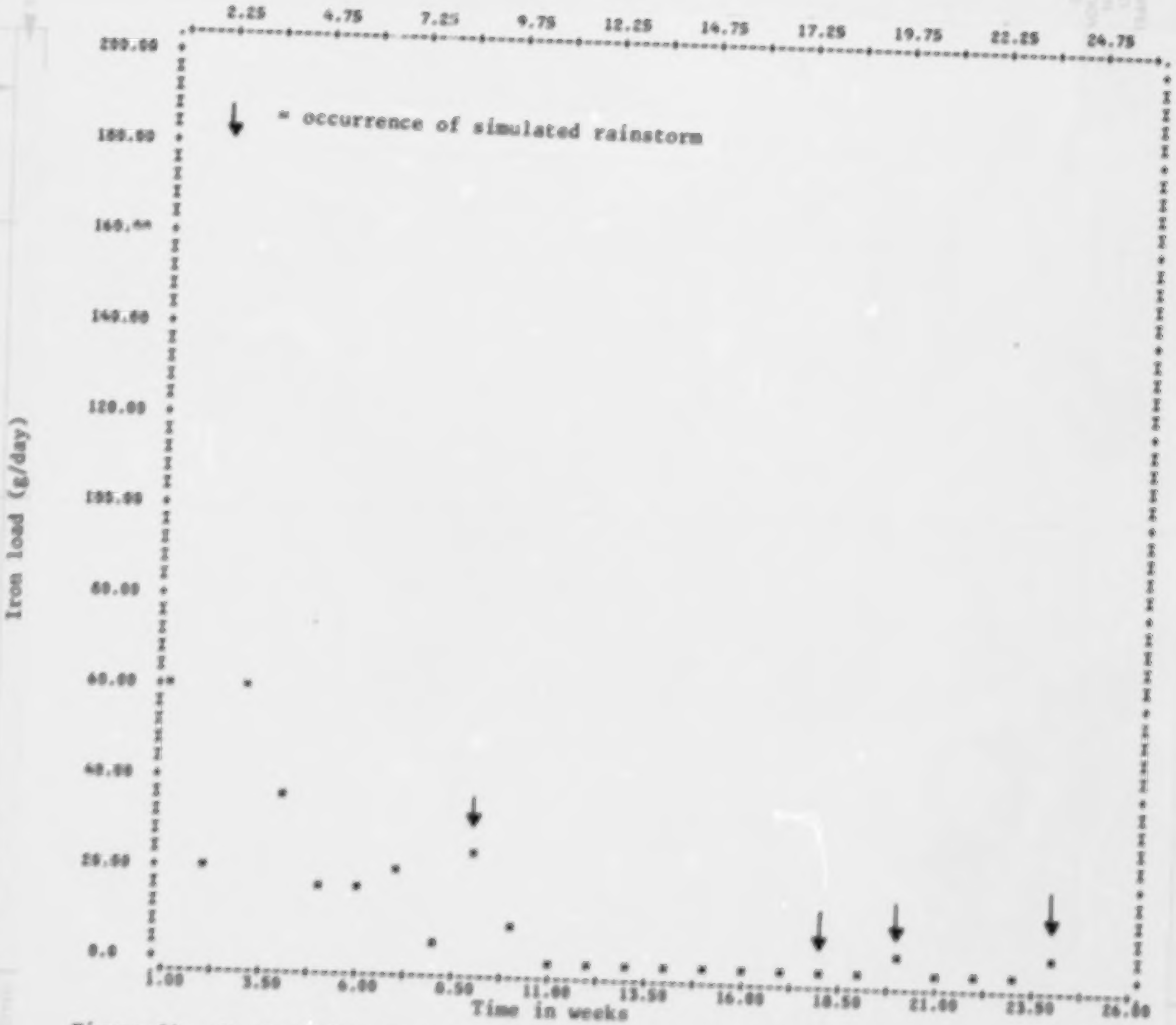

Figure 61. Graph of iron load as a function of time for Column 17 (5.0-cm (2-in) neutralizing layer).

comparison, leachate from columns containing sludge and fertilizer layered over the lime-treated waste (columns 7 and 8) also had a maximum mean iron loading after the first simulated storm event. This maximum mean iron loading was 126 g/day, approximately 1.5 times greater than that noted above from columns containing the materials incorporated. The mean iron loads noted in the leachates from columns containing the layered materials ranged from 126 g/day to less than 2 g/day and averaged approximately 25 g/day (Figure 55). This represented an overall decrease of approximately 55 percent when compared to the untreated mine waste control columns (columns 1 and 2). The rate of decrease in iron load with respect to time for columns 5 and 6 (incorporated amendments) was approximately 1.9 times greater than the same rate for columns 7 and 8 (layered materials).

Leachates from columns containing cover soil layered over the mine waste (columns 9 and 10) also had reduced iron loads. The mean iron loads in the leachate from columns containing 0.5 m (1.5 ft) of cover soil decreased from an initial maximum load of 122 g/day to a minimum of less than 2 g/day (Figure 56), corresponding to an overall decrease of approximately 50 percent when compared to the untreated mine waste (columns 1 and 2). Likewise, use of a 0.61-m (2-ft) layer of cover soil positioned over 1.2 m (4 ft) of the mine waste (columns 11 and 12) resulted in an overall decrease of approximately 60 percent (Figure 57) in the iron load of the resulting leachate when compared to that from the mine waste alone (columns 1 and 2). The most significant decrease in leachate iron load was noted when a 1.2-m (4-ft) layer of cover soil was placed over the mine waste. This resulted in an overall decrease in the iron load of the leachate of approximately 71 percent. This correlated to mean iron loads ranging from a maximum of about 92 g/day observed initially during the study period, to a minimum of less than 1 g/day (Figure 58). Mean iron loads from these columns averaged approximately 16 g/day and decreased steadily. The rates of decrease in the mean iron loading over time for columns containing 0.5 m (1.5 ft) of cover soil (columns 9 and 10) or 0.61 m (2 ft) of cover soil (columns 11 and 12) averaged between one-third to three-fourths the rate of decrease for columns containing 1.2 m (4 ft) of cover soil positioned over the mine waste (columns 13 and 14).

Iron loads of the leachates from columns containing a neutralizing layer between the mine waste and cover soil material (columns 15 and 17) also varied compared to the control (column 16). When compared to that of column 16 (no neutralizing layer) the mean iron loads of these experimental columns showed overall increases of approximately 45 to 48 percent. These increases may be in part a result of the decreased flow rates (Appendix A) noted in these two columns relative to column 16. As a result of the decreased flows, there may have been more build-up of pyritic oxidation products, such as iron, so that greater concentrations of iron were noted in the leachates. This conforms to Nordstrom and Dagenhart's (1979) view of metal leaching at the Contrary Creek site. They felt that extended dry periods with little or no flushing of the pyritic refuse resulted in a build-up of oxidation products which ultimately resulted in high concentrations of these products in the leachate. The increases and deviations in leachate metal loading data from these experimental columns compared to the control appear to be significantly greater than the corresponding deviations observed in the acidity and specific conducting data discussed earlier. It is therefore suggested that these trends may not persist or be indicitive of what would occur over a longer study period.

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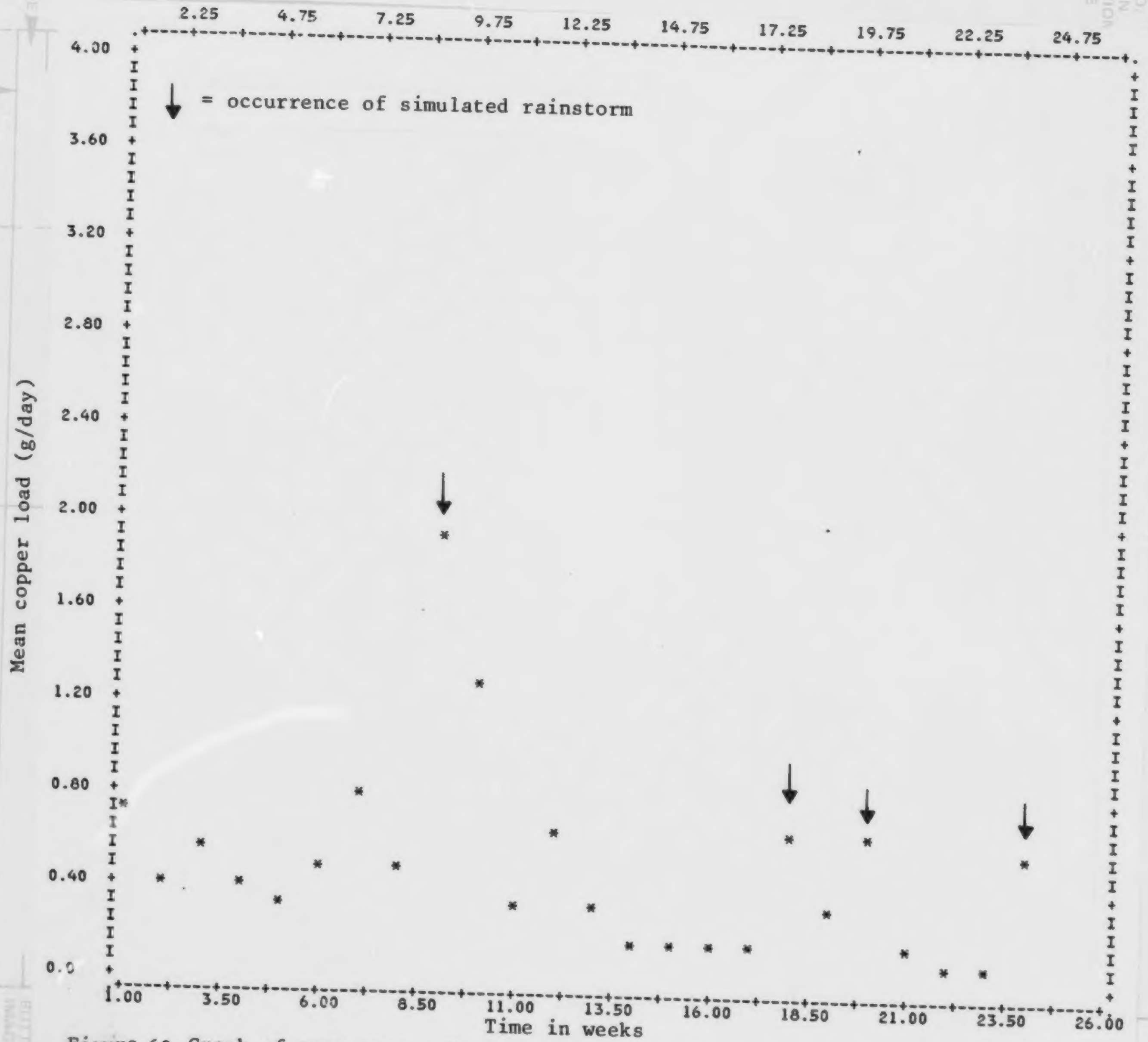

Figure 62. Graph of mean copper load as a function of time for Columns 1 and 2 (mine waste controls).

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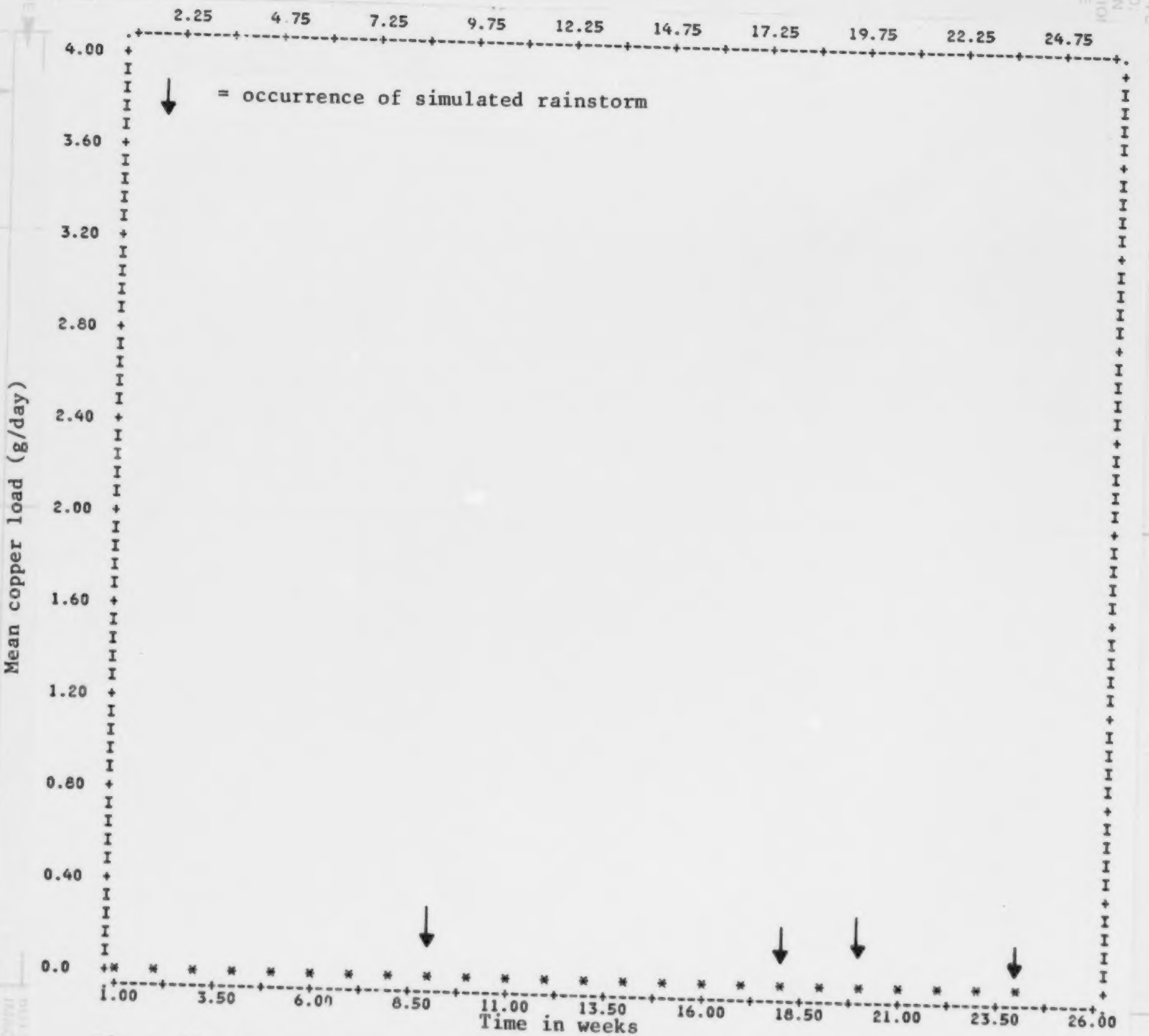

Figure 63. Graph of mean copper load as a function of time for Columns 3 and 4 (cover soil controls).

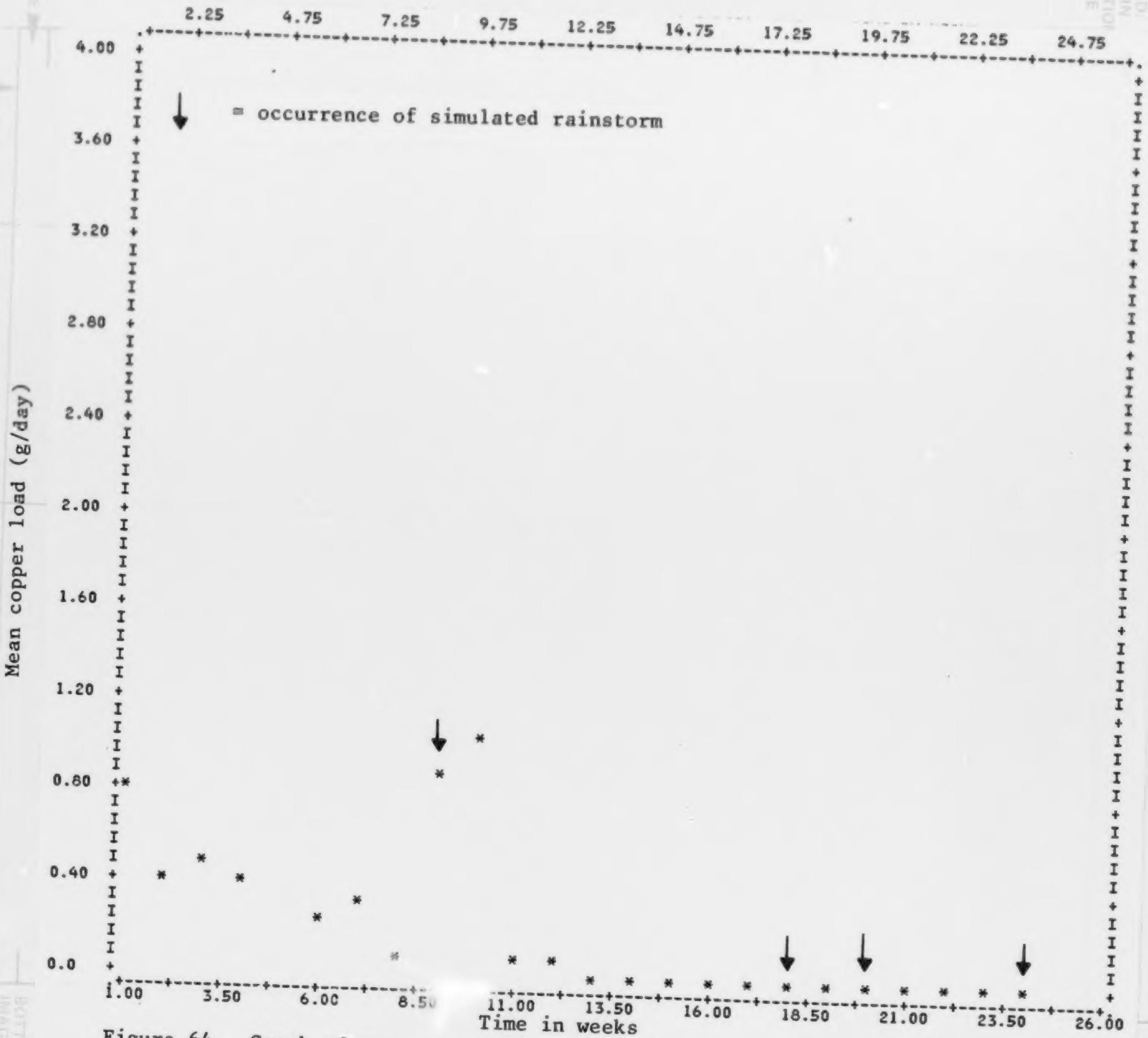

Figure 64. Graph of mean copper load as a function of time for Columns 5 and 6 (incorporated amendments).

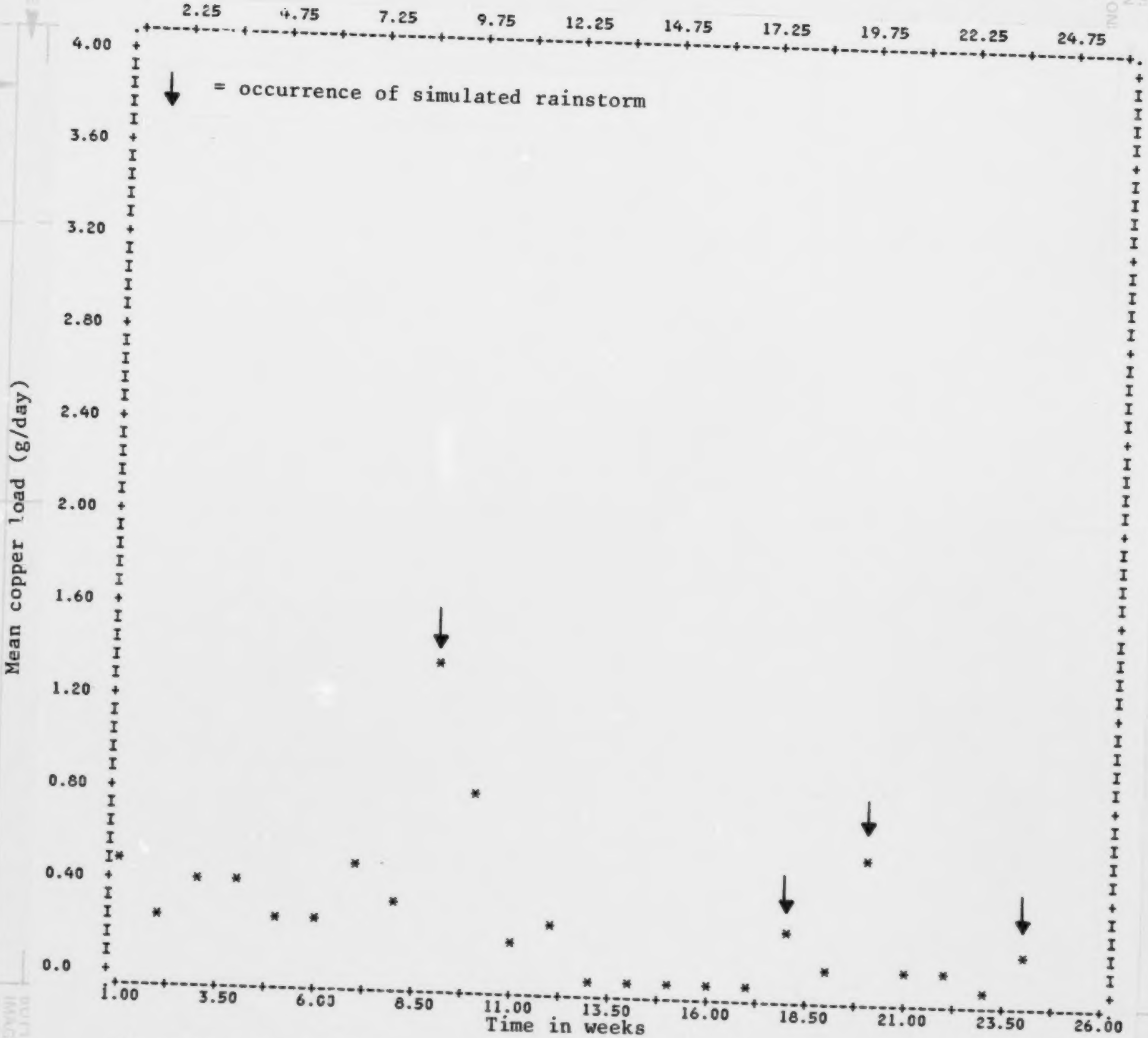

Figure 65. Graph of mean copper load as a function of time for Columns 7 and 8 (layered amendments).

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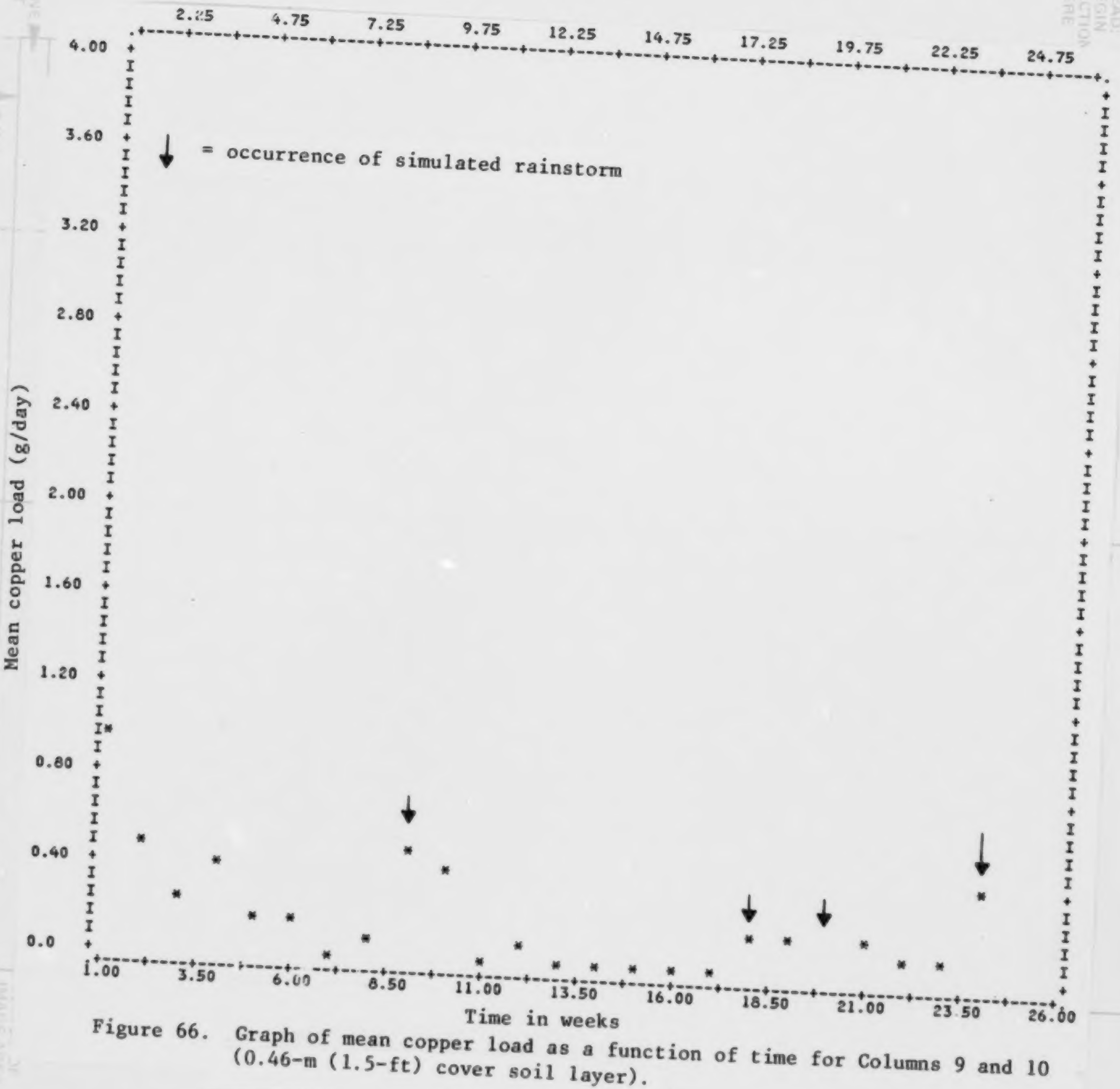

Figure 66. Graph of mean copper load as a function of time for Columns 9 and 10 (0.46-m (1.5-ft) cover soil layer).

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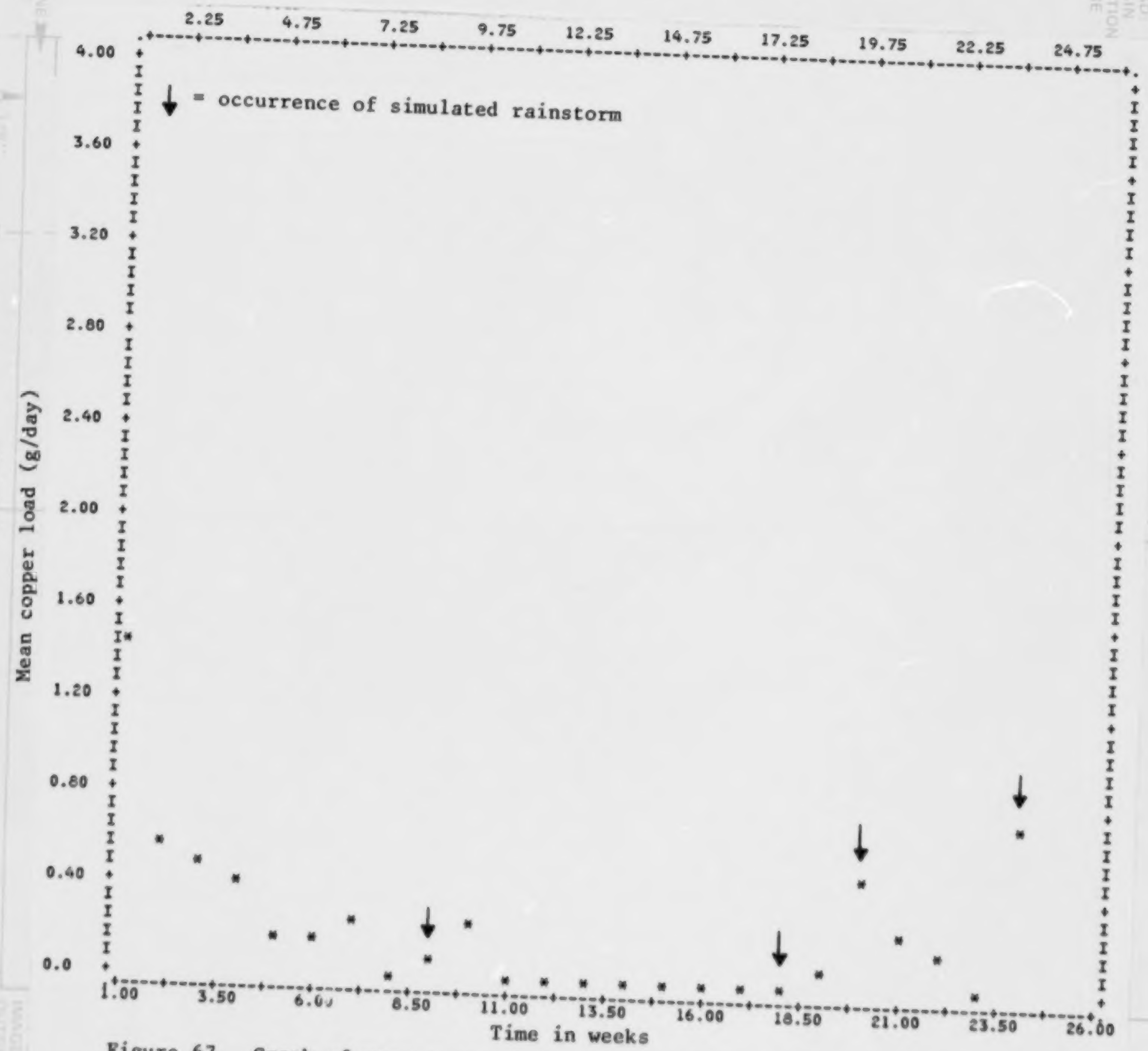

Figure 67. Graph of mean copper load as a function of time for Columns 11 and 12 (0.61-m (2-ft) cover soil layer).

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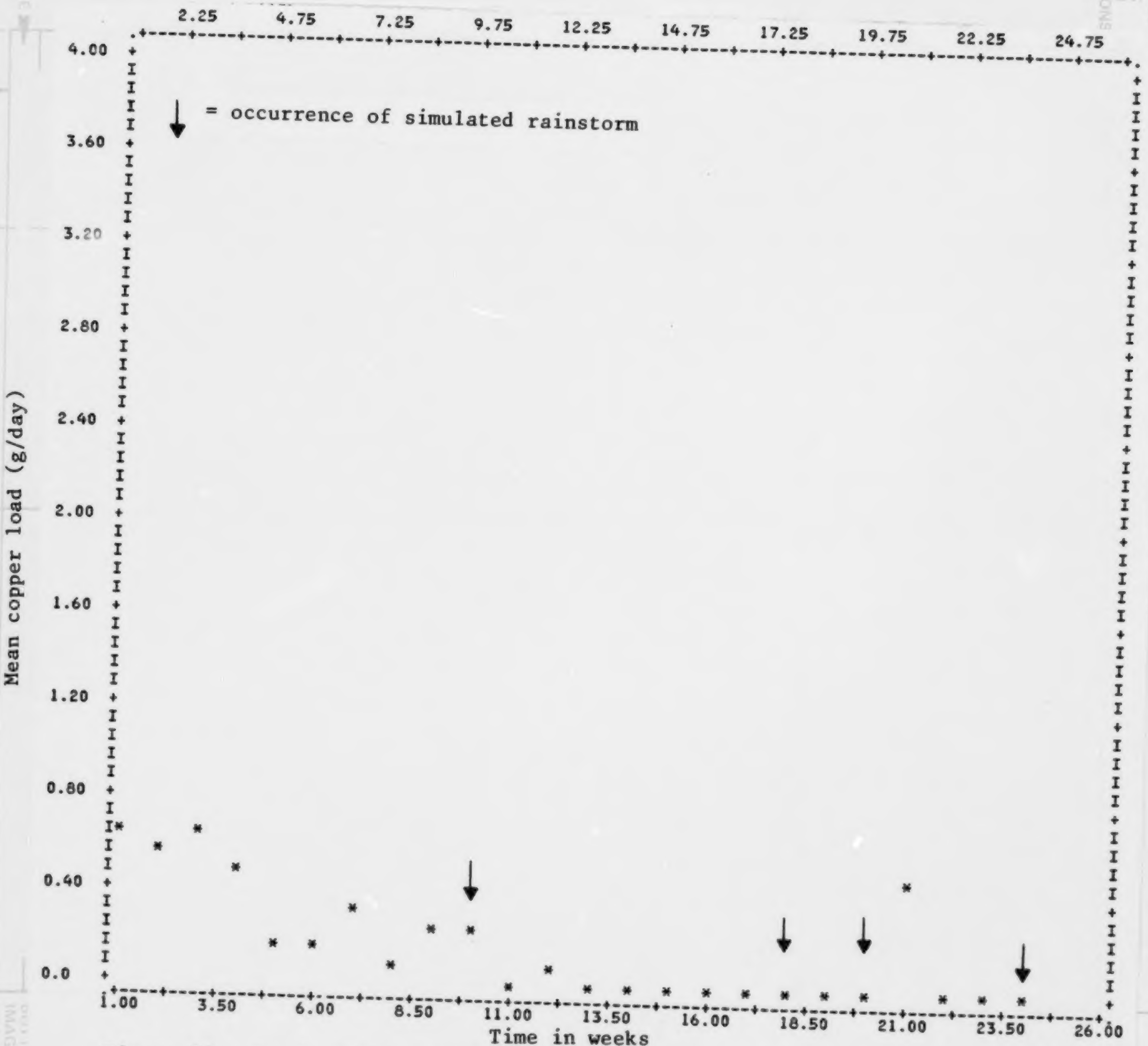

Figure 68. Graph of mean copper load as a function of time for Columns 13 and 14 (1.2-m (4-ft) cover soil layer).

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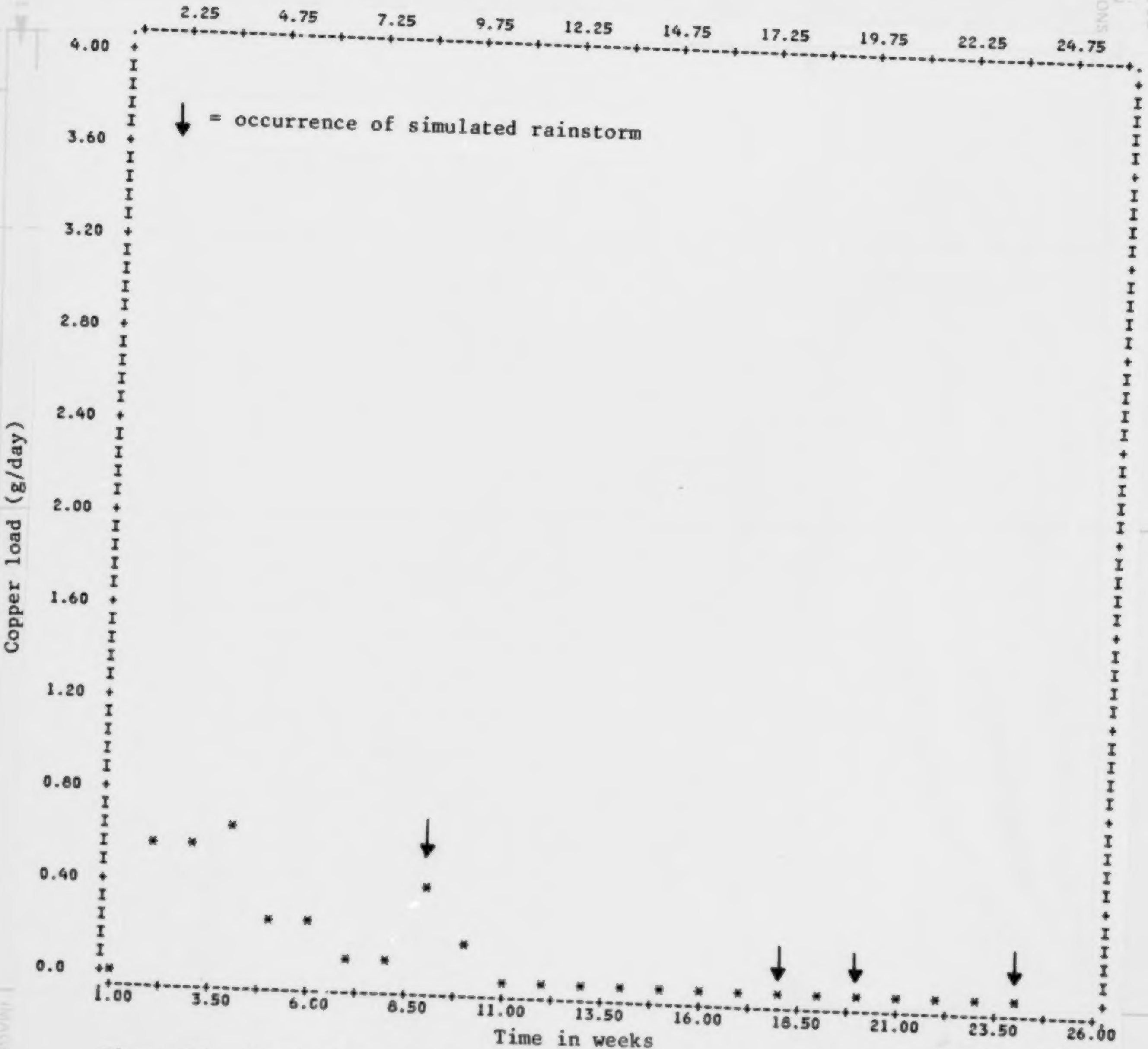

Figure 69. Graph of copper load as a function of time for Column 15 (2.5-cm (1-in) neutralizing layer).

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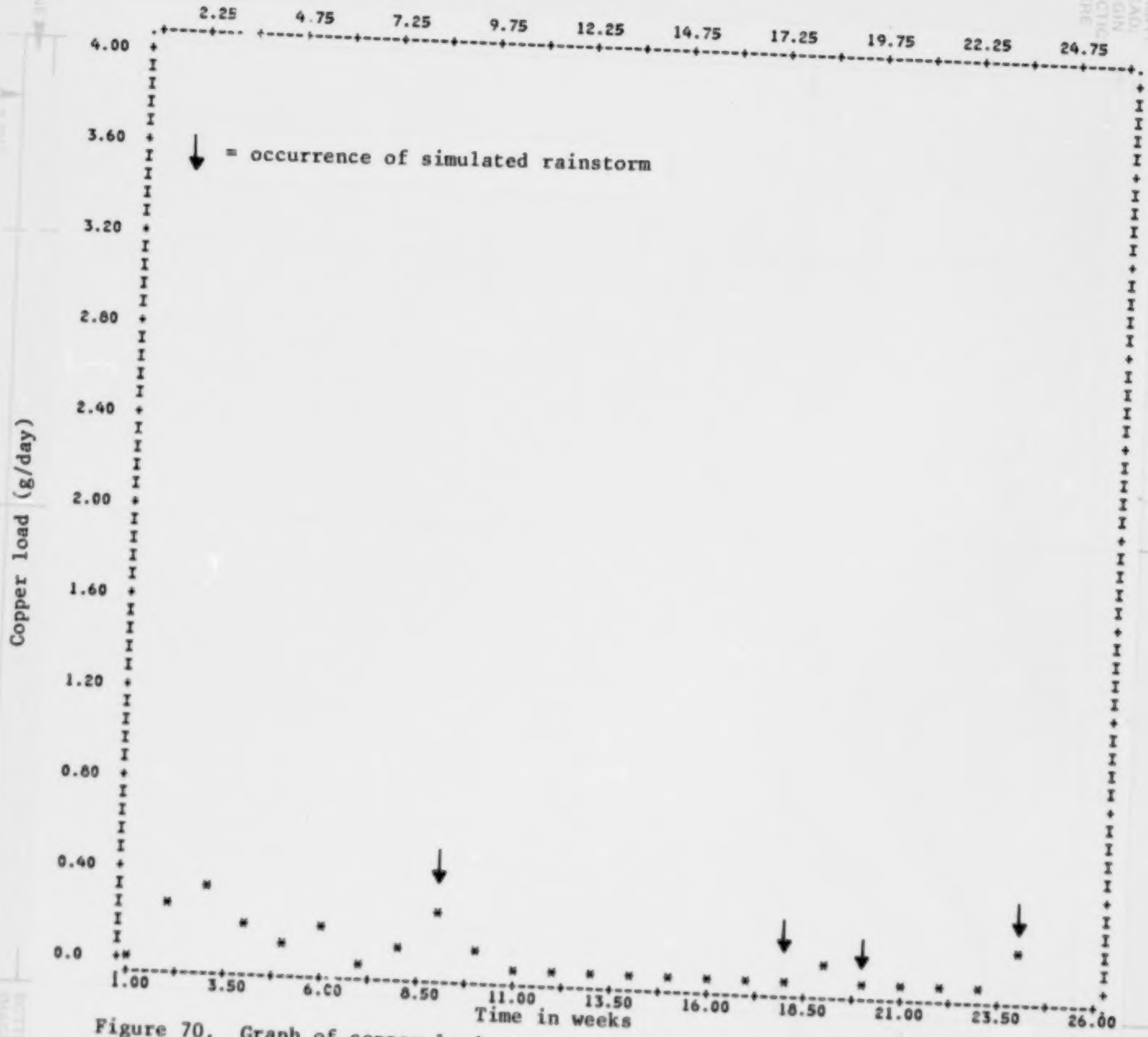

Figure 70. Graph of copper load as a function of time for Column 16 (cover soil/mine waste control).

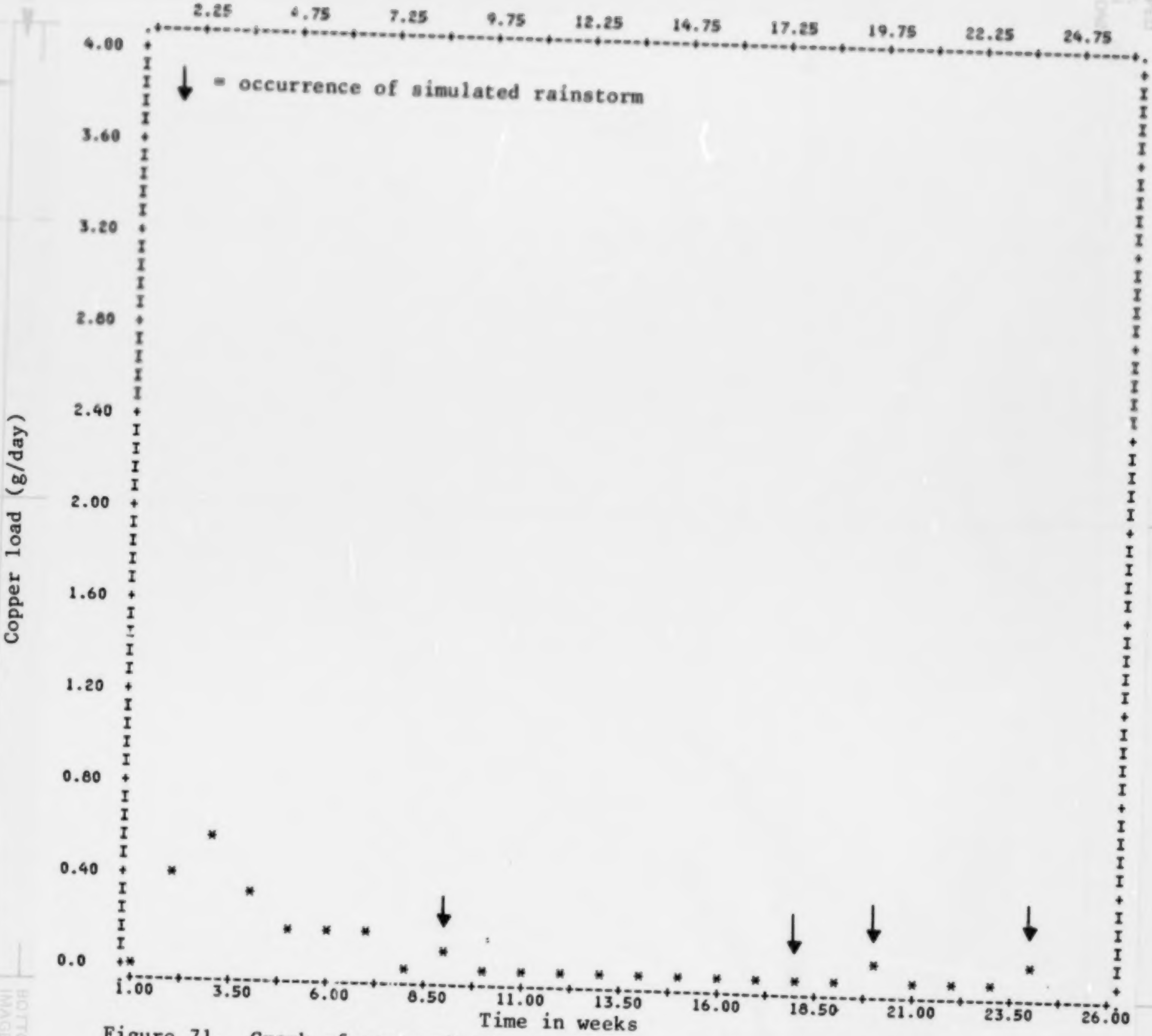

Figure 71. Graph of copper load as a function of time for Column 17 (5.0-cm (2-in) neutralizing layer).

Copper--

The mean copper loads resulting from each treatment technique were also graphed as a function of time as shown in Figures 62 through 71. Copper loads in the leachate from columns containing the untreated pyrite mine waste (columns 1 and 2) averaged approximately 0.50 g/day and ranged from a maximum of 1.9 g/day to a minimum of less than 0.1 g/day (Figure 62). As with iron, maximum copper loads corresponded with increased water applied to simulate rainfall events. Copper loads in the leachate from the soil control columns (columns 3 and 4) averaged approximately 0.001 g/day as in the case of a normal top soil (Figure 63).

Copper loads in the leachate from the columns containing the amendments incorporated into the upper strata of the mine waste (columns 5 and 6) ranged from 1 to less than 0.01 g/day, averaging 0.21 g/day (Figure 64). This corresponded to an overall decrease in the mean copper load of 58 percent when compared to the control columns. In comparison, a 44-percent decrease in the mean copper load of the leachate from columns containing the sludge and fertilizer layered over the lime-treated waste (columns 7 and 8) was noted. Copper loads from these columns ranged from approximately 1.4 to approximately 0.02 g/day, averaging 0.28 g/day (Figure 65). In comparing the effects of the two amendment configurations upon leachate copper loads, it was observed that the rate of decrease in copper load over time was 1.6 times greater when the materials were incorporated as compared to that when they were layered.

In comparison to the mine waste control columns, a 0.5-m (1.5-ft) cover soil layer placed over the mine waste resulted in a 56-percent decrease in the mean copper load of the leachate. A maximum loading of approximately 1.0 g/day was noted initially with a minimum of less than 0.02 g/day noted ten weeks later (Figure 66). An average copper loading of 0.22 g/day was noted in the leachate from these columns throughout the study period. Likewise, columns containing 0.61 m (2 ft) (columns 11 and 12) and 1.2 m (4.0 ft) (columns 13 and 14) of cover soil layered over the mine waste also showed reduced copper loads when compared to the control columns (columns 1 and 2). Columns 11 and 12 showed an overall decrease of 54 percent (Figure 67) in the mean copper load of the leachate when compared to the control columns. As noted with other variables discussed earlier, very little difference was observed in the copper loads of leachate from columns containing a 0.5-m (1.5-ft) layer of cover soil and columns containing a 0.61-m (2.0-ft) layer of cover soil. Only a slightly higher overall decrease was noted from columns containing a 2.1-m (4-ft) layer of cover soil placed over the mine waste. Mean copper loads in leachates from these columns ranged from about 0.7 g/day to less than 0.01 g/day, averaging approximately 0.18 g/day (Figure 68). This corresponded to an overall decrease in the mean copper load of 64 percent when compared to the mine waste with no cover soil (columns 1 and 2). The slight difference in overall decreases is also evident in the rates of decrease in mean copper loads over time for these columns. Use of a 1.2-m (4-ft) layer resulted in a rate of decrease in mean copper load over time that was approximately 1.4 times that when a 0.61-m (2-ft) layer of cover soil was used and 1.6 times that noted when a 0.5-m (1.5- ft) layer of cover soil was used.

Copper loads of the leachate from columns 15 and 17, containing 2.5-cm

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(1-in) and 5-cm (2-in) lime layers respectively, showed an increase relative to the control (column 16) as in the case of iron loads previously discussed. An overall increase in copper load of approximately 110 percent was noted when a 2.5-cm (1-in) layer of lime was used (Figure 69) as compared to an increase of approximately 39 percent when a 5-cm (2-in) layer of lime was used (Figure 71). Overall rates of increase in copper loads over the study period were approximately 2.0 times that of the control (column 16) for both columns 15 and 17. This is analogous to the iron data obtained from these columns and also may be explained at least in part by the differences in flow rate and may or may not be indicitive of what would occur over a longer study period.

Zinc--

As with copper and iron, the mean zinc loads for each treatment technique were determined and plotted as a function of time as shown in Figures 72 through 81. Mean zinc loads noted in the leachate from the untreated pyrite control columns (columns 1 and 2) ranged from a maximum mean load of approximately 1.6 g/day occurring after a storm event to a minimum mean load of approximately 0.2 g/day (Figure 72). This resulted in an average load of 0.55 g/day. Mean zinc loads of the leachate from the cover soil control columns (3 and 4) remained low and negligible throughout the study period (Figure 73).

In comparison to the mine waste control columns, the incorporation of lime, sludge, and fertilizer into the top 15.2 cm (6 in) of the mine waste resulted in an overall decrease of 67 percent in the mean zinc load of the leachate. A maximum mean zinc loading of almost 1 g/day was observed after the first simulated storm event with a minimum of 0.007 g/day noted eight weeks later (Figure 74). An average loading of 0.18 g/day was noted throughout the study period. In comparison, leachate from columns containing sludge and fertilizer layered over the lime-treated waste (columns 7 and 8) also had the maximum loading after the first simulated storm event. This maximum mean zinc loading was approximately 1.1 g/day, very similar to that noted when the materials were incorporated. A minimum mean zinc loading of 0.01 g/day was noted with an average loading of 0.27 g/day occurring throughout the study period (Figure 75). This average is 1.5 times greater than that noted when the materials were incorporated (columns 5 and 6). In addition, the rate of decrease in zinc load over time for the columns containing the materials incorporated was approximately four times that for the columns containing the materials layered.

Leachates from columns containing cover soil layered over the mine waste also had reduced zinc loads. The mean zinc loads of leachate from columns containing 2.1 m (4 ft) of cover soil decreased from an initial maximum load of 1.1 g/day to a minimum load of less than 0.01 g/day (Figure 78), corresponding to an overall decrease of 69 percent when compared to the untreated mine waste control columns (columns 1 and 2). This compared to overall decreases of 56 percent and 53 percent noted in leachate from columns containing 0.5 m (1.5 ft) and 0.61 m (2 ft) layers of cover soil over the mine waste, respectively, (Figures 76 and 77).

The mean zinc loads of leachates from columns containing a neutralizing layer placed between the mine waste and cover soil material (columns 15 and

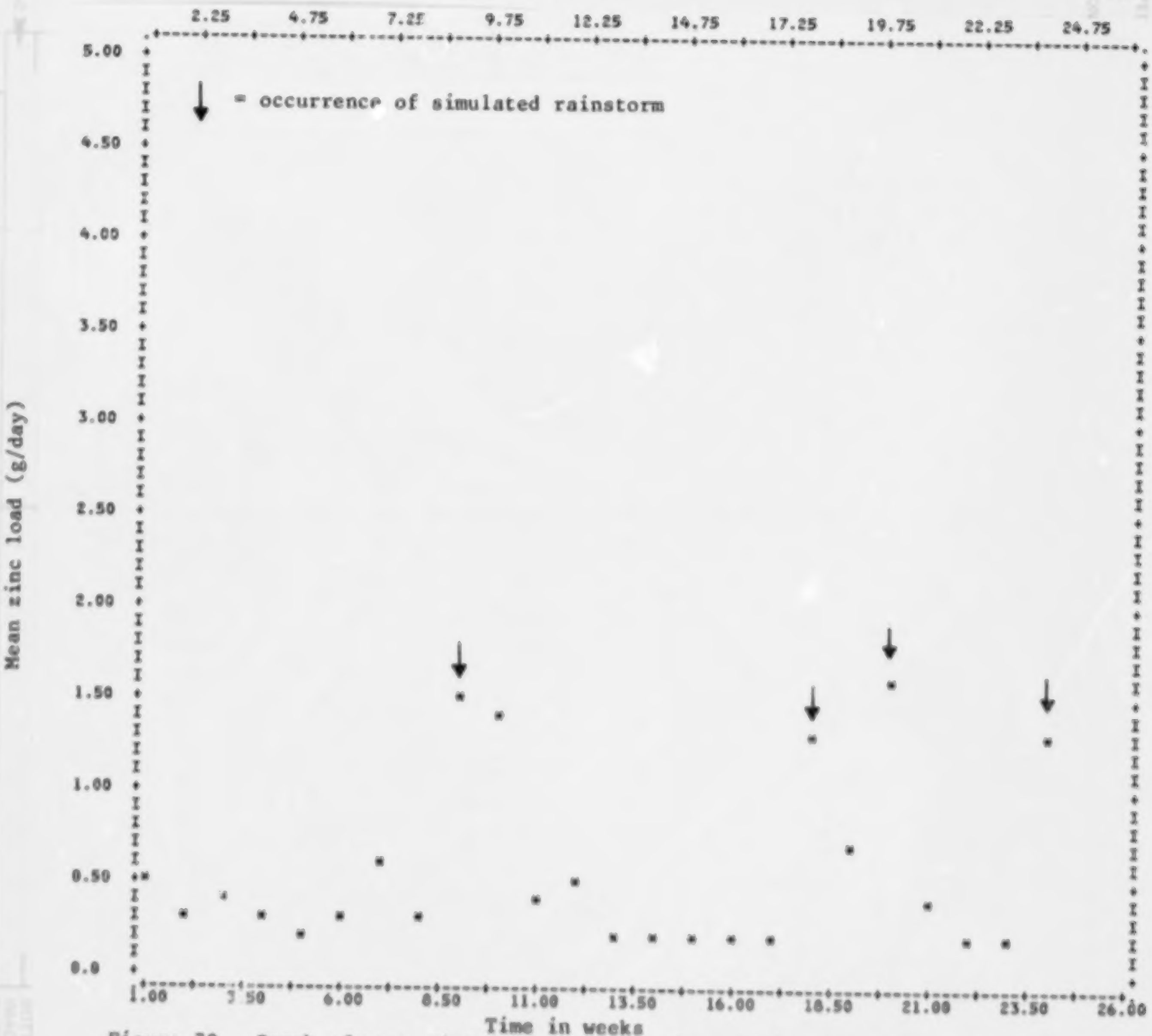

Figure 72. Graph of mean zinc load as a function of time for Columns 1 and 2 (mine waste controls).

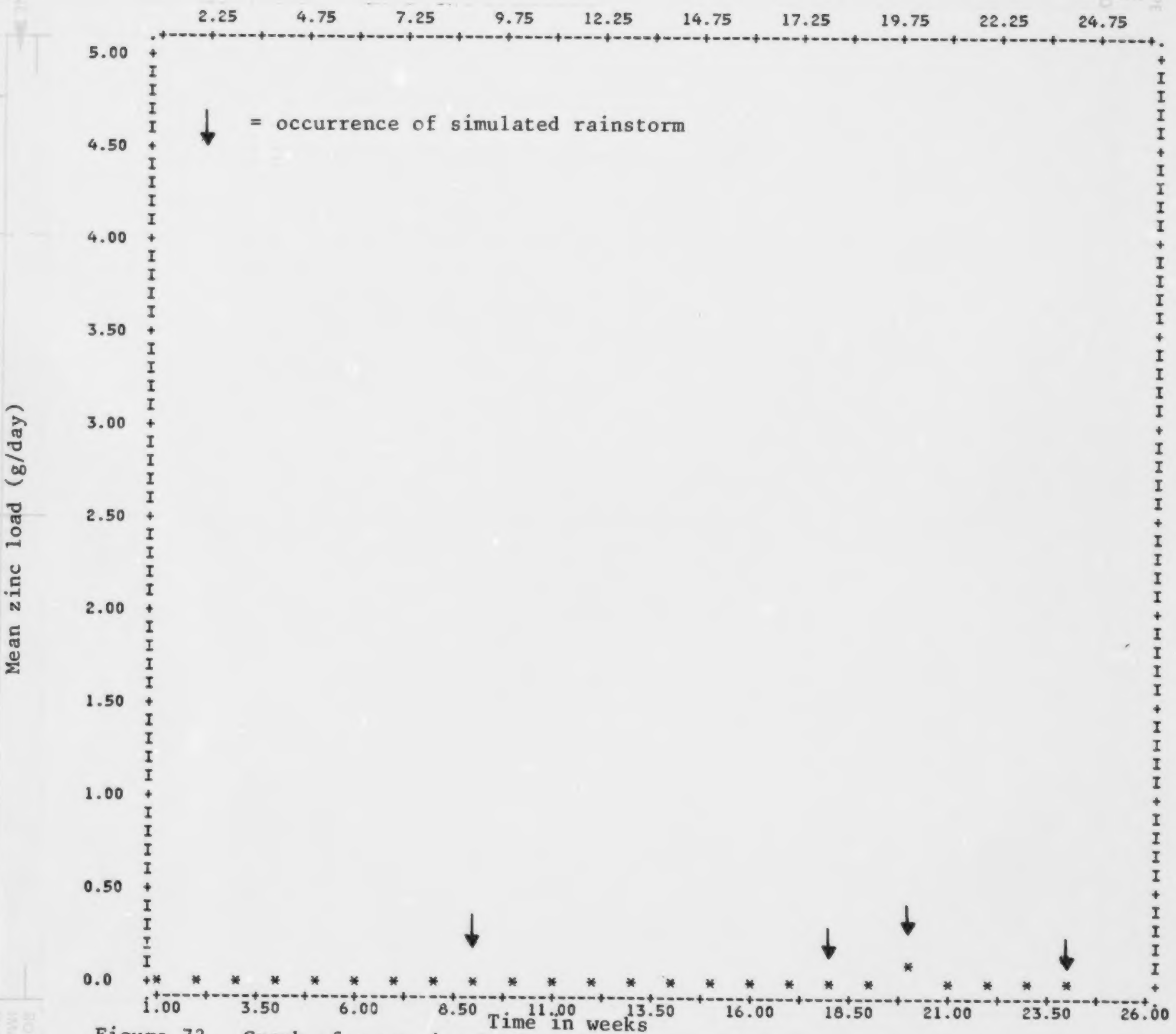

Figure 73. Graph of mean zinc load as a function of time for Columns 3 and 4 (cover soil controls).

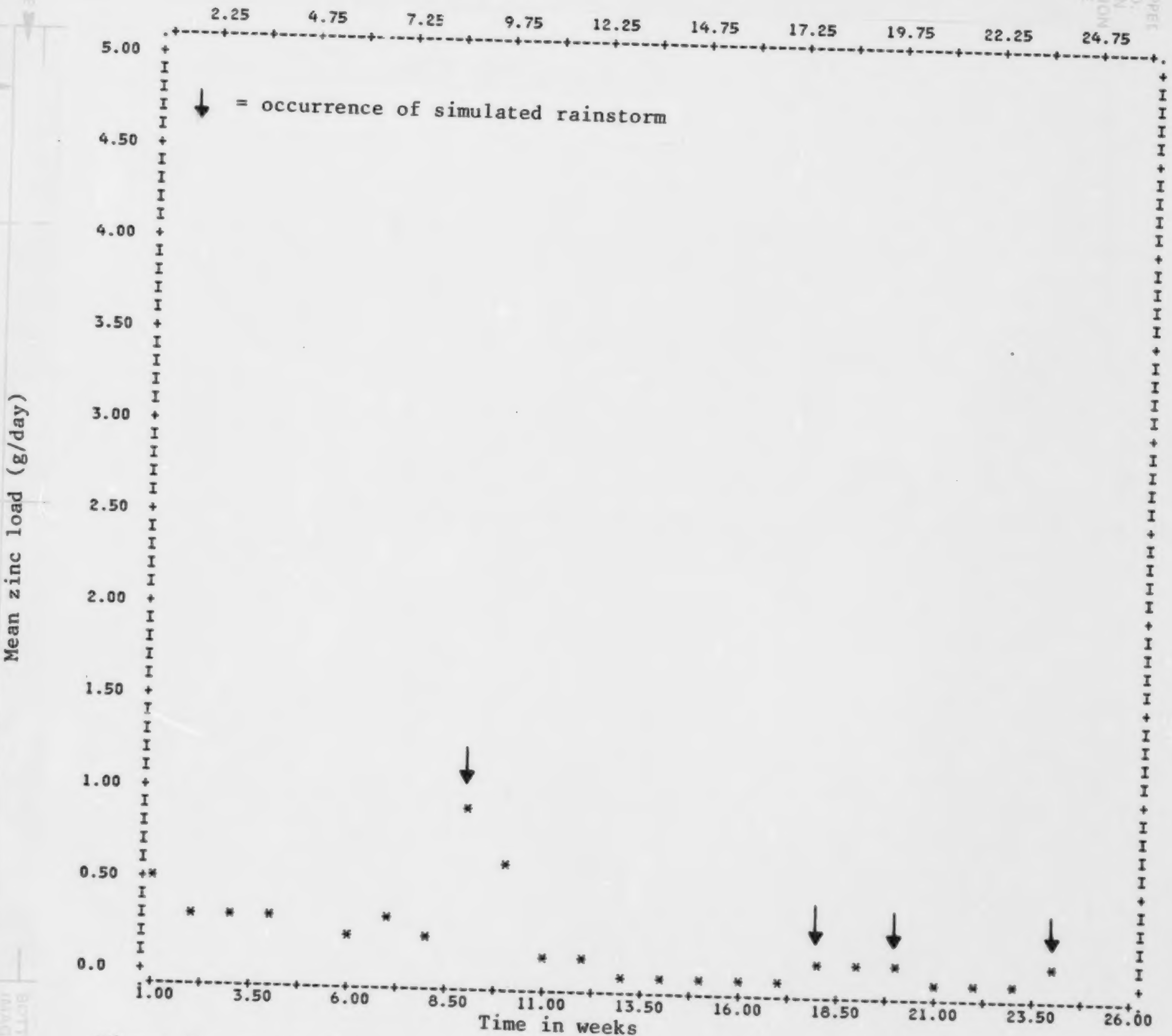

Figure 74. Graph of mean zinc load as a function of time for Columns 5 and 6 (incorporated amendments).

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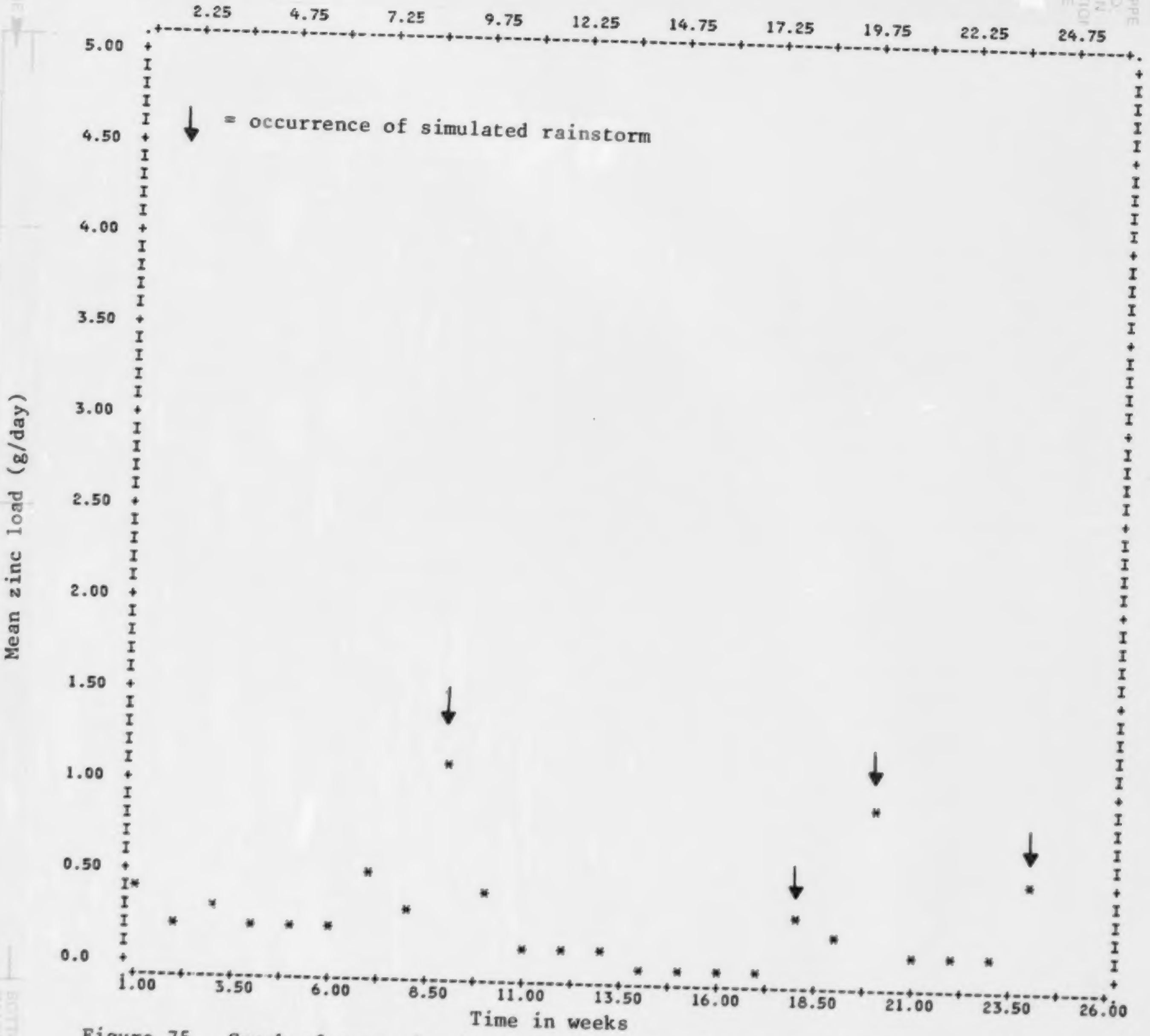

Figure 75. Graph of mean zinc load as a function of time for Columns 7 and 8 (layered amendments).

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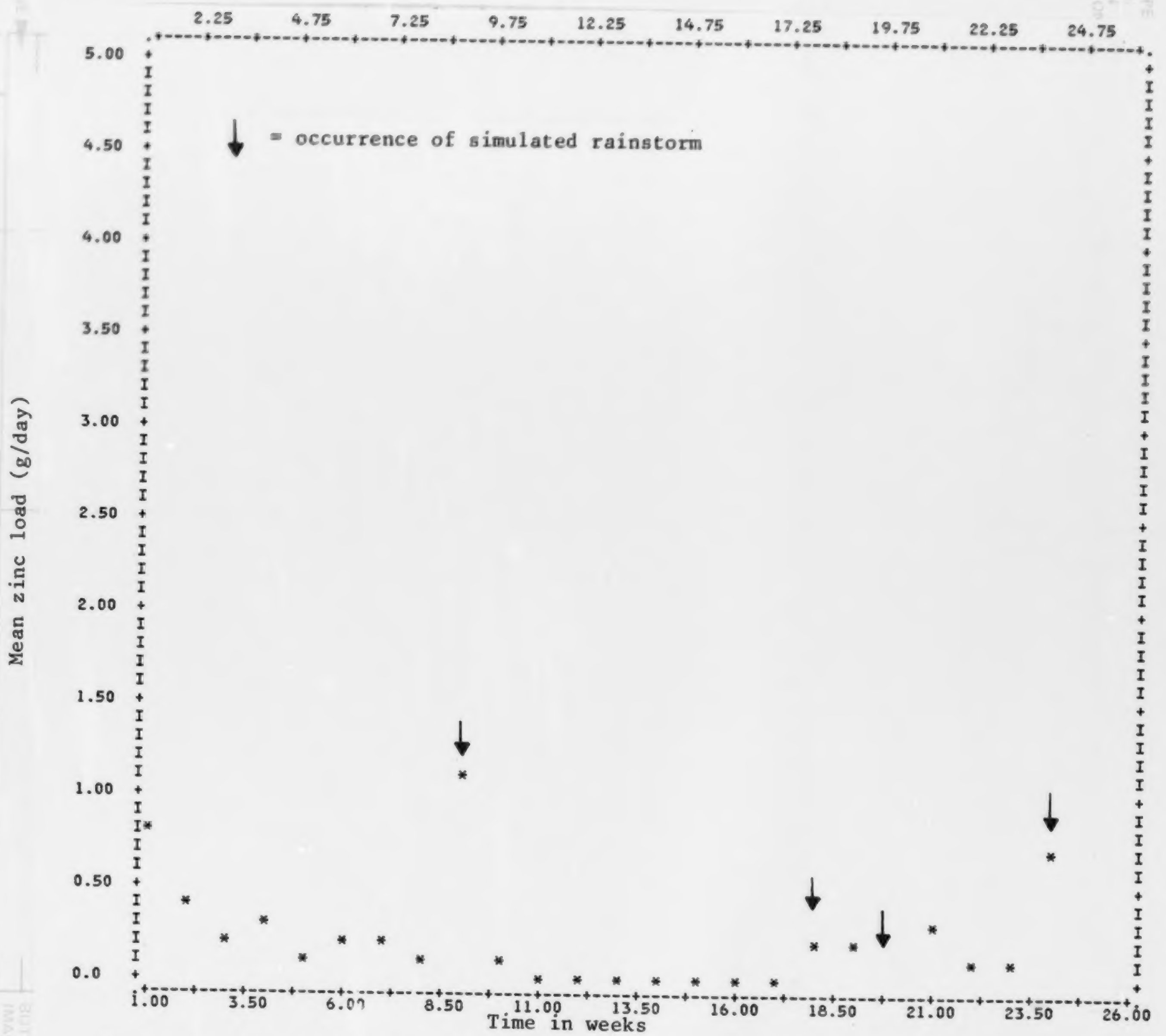

Figure 76. Graph of mean zinc load as a function of time for Columns 9 and 10 (0.46-m (1.5-ft) cover soil layer).

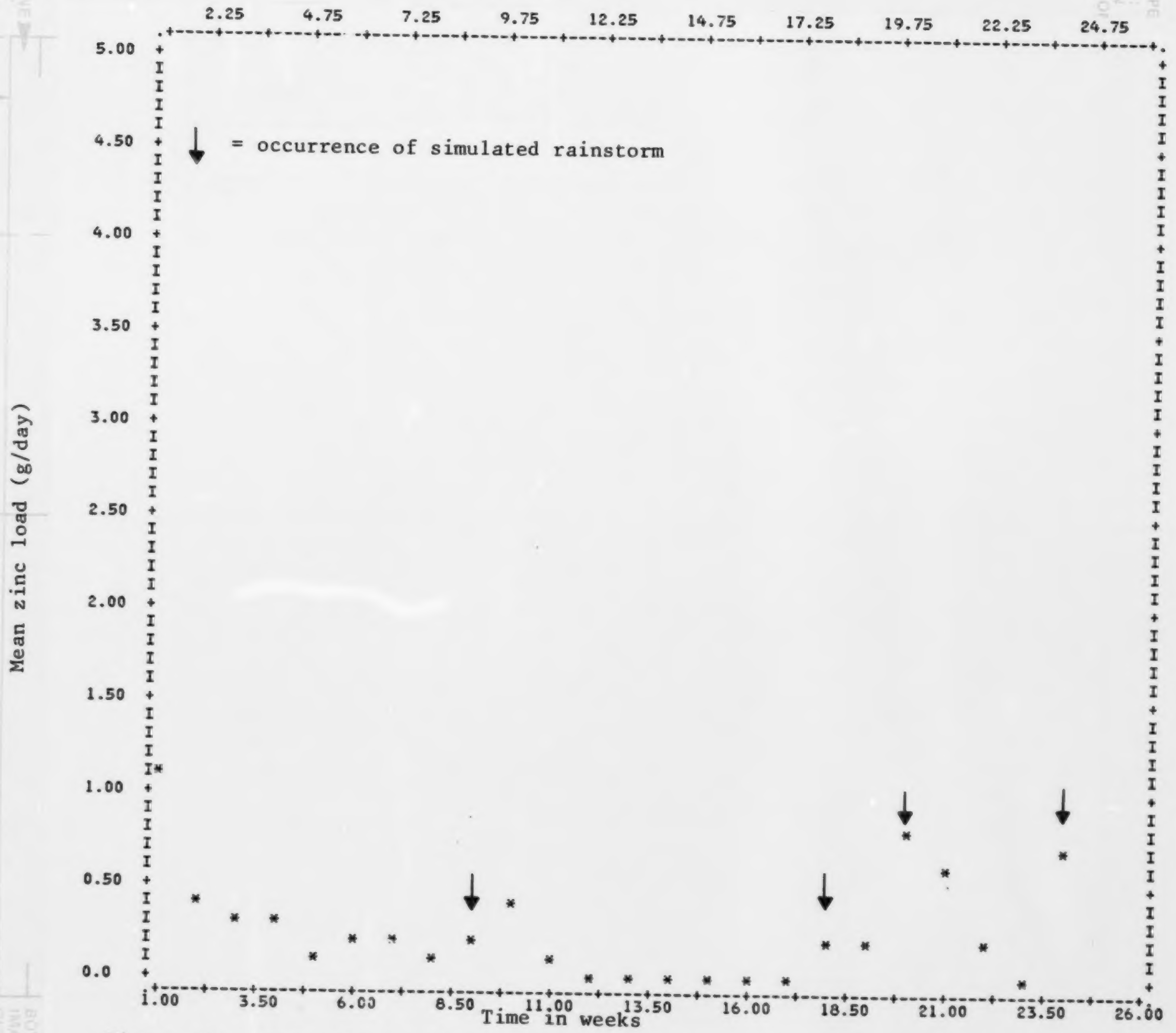

Figure 77. Graph of mean zinc load as a function of time for Columns 11 and 12 (0.61-m (2-ft) cover soil layer).

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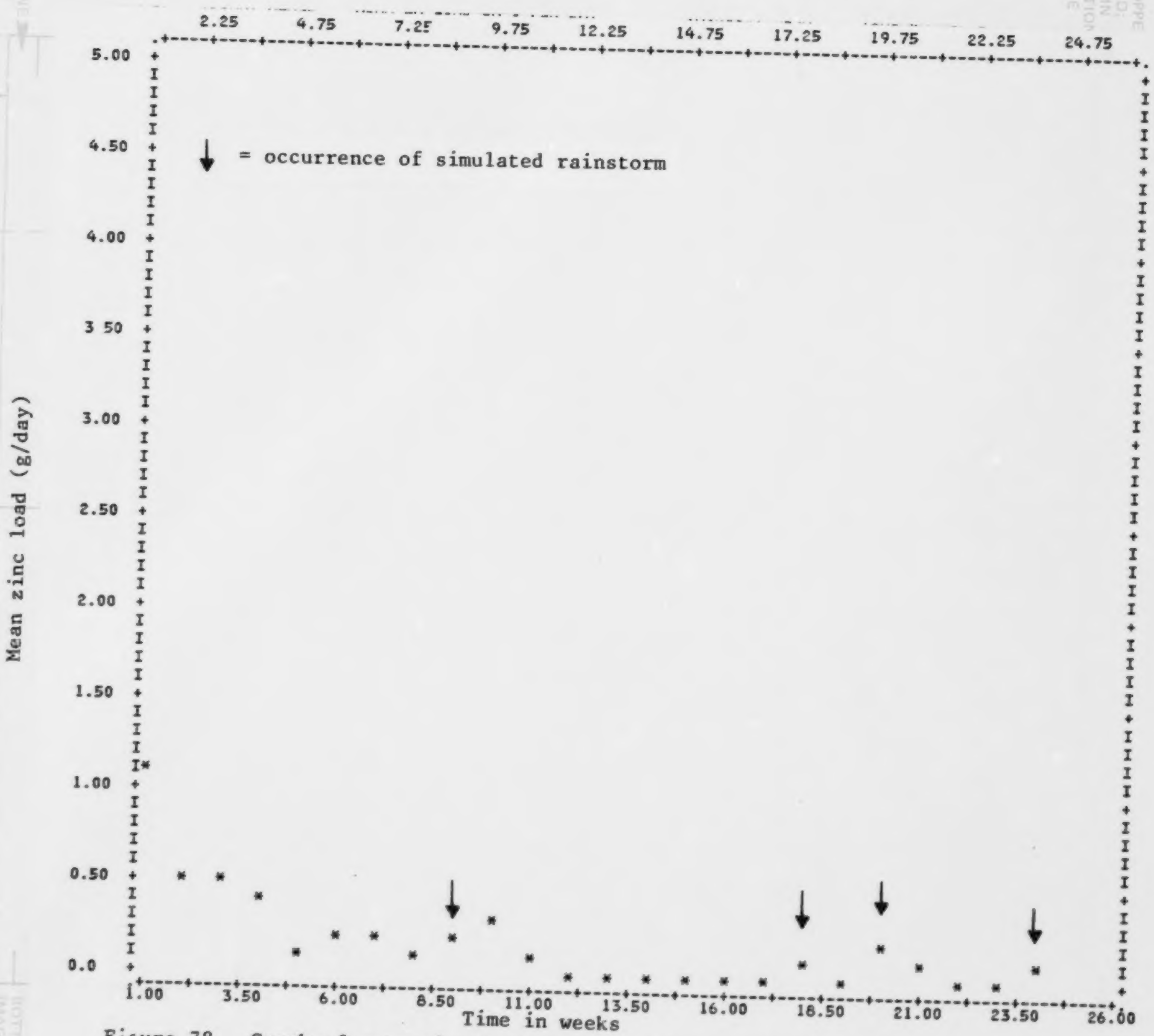

Figure 78. Graph of mean zinc load as a function of time for Columns 13 and 14 (1.2-m (4-ft) cover soil layer).

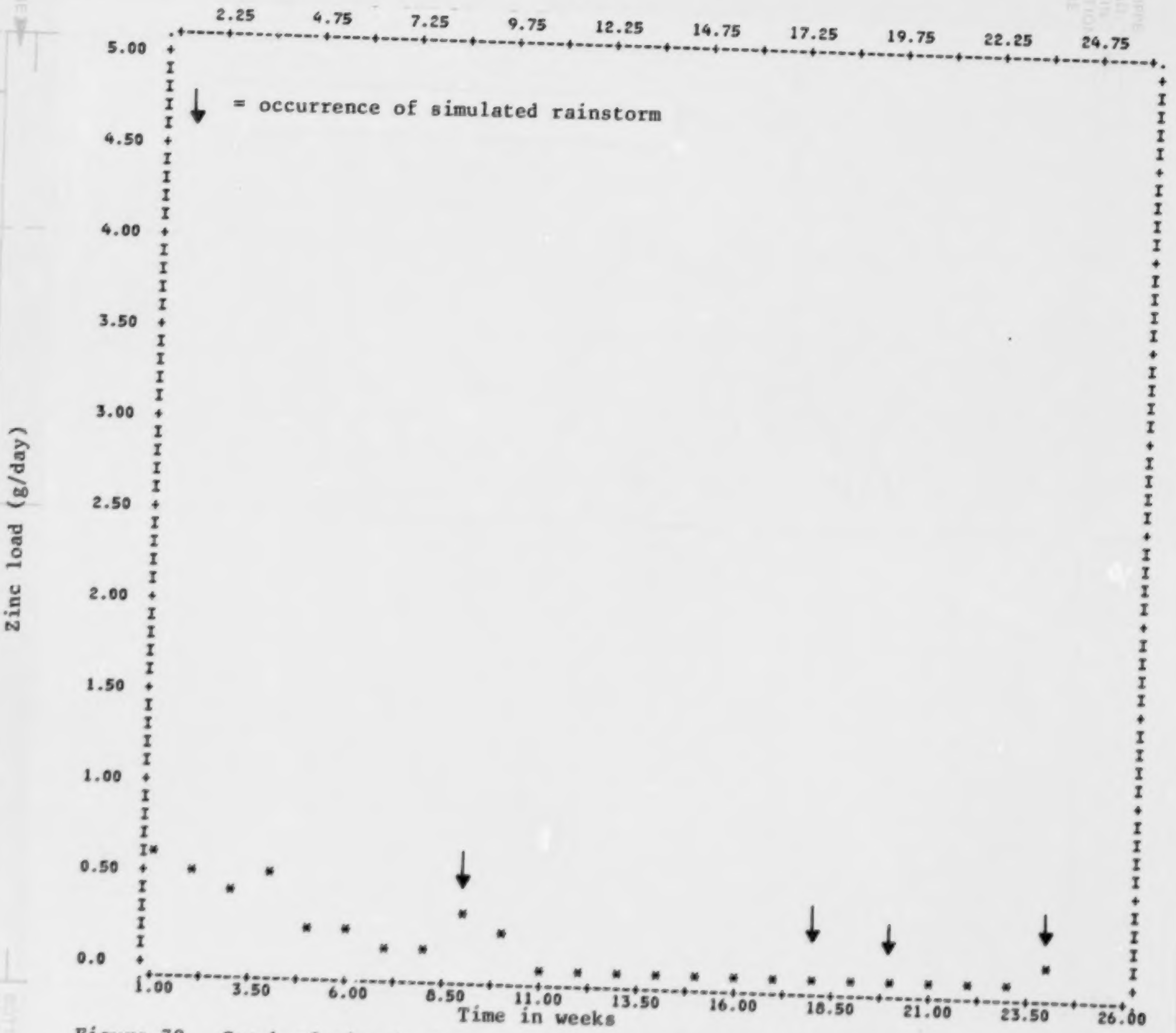

Figure 79. Graph of zinc load as a function of time for Column 15 (2.5-cm (1-in) neutralizing layer).

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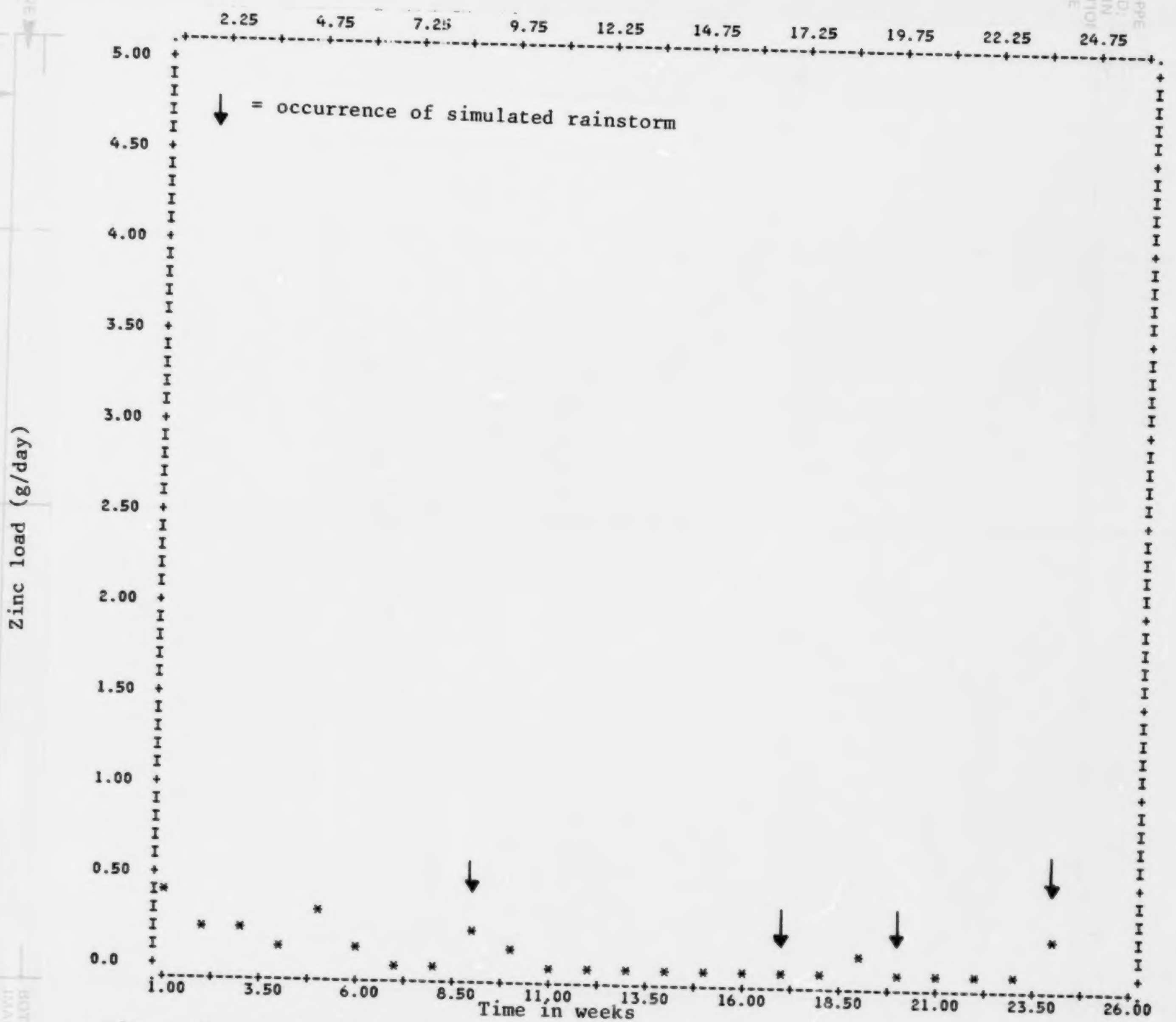

Figure 80. Graph of zinc load as a function of time for Column 16 (cover soil/mine waste control).

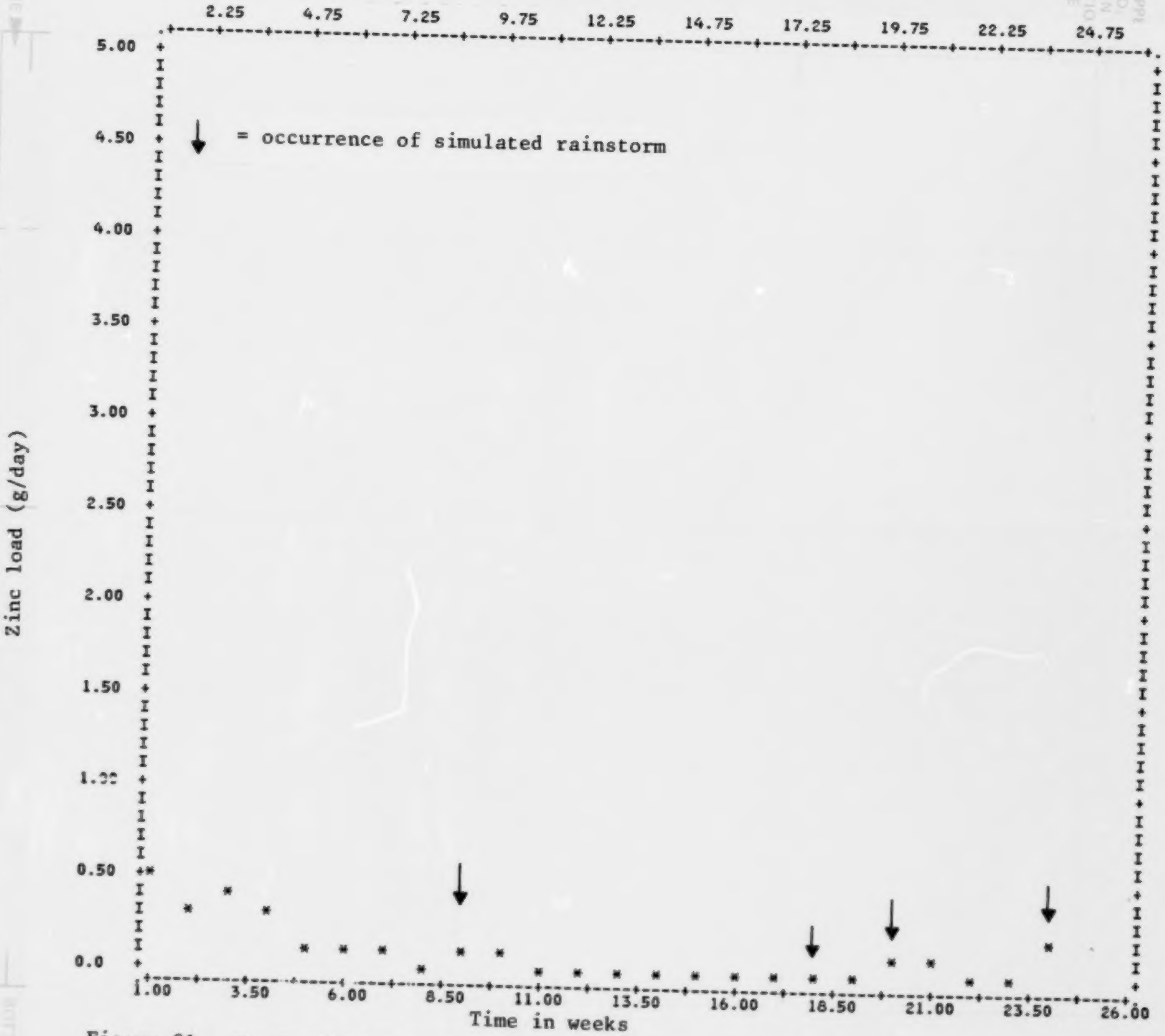

Figure 81. Graph of zinc load as a function of time for Column 17 (5.0-cm (2-in) neutralizing layer).

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17) showed increases of approximately 51 percent when a 2.5-cm (1-in) lime layer was used and 43 percent when a layer twice as thick was used (Figures 79 and 81). This data corresponds with increased metal loads noted earlier and may be due to decreased flows and the build-up of pyrite oxidation products as discussed earlier. There deviations were also greater than those seen in the specific conductivity and acidity data, and may not persist over a longer study period.

Lead--

The mean lead loads in the leachates from all the columns were also plotted as a function of time as shown in Figures 82 through 91. Overall, lead was present in the leachate in very low quantities as compared to iron, copper, and zinc. The average leachate mean lead load from the mine waste control columns (columns 1 and 2) was approximately 0.012 g/day and showed little variability throughout the study period (Figure 82). Lead loads present in the leachate from the cover soil control columns were also low (Figure 83) and typical for a normal topsoil.

These loads compared to an average lead load of 0.001 g/day (Figure 84) from columns containing the sludge, lime, and fertilizer incorporated into the upper strata of the mine waste (columns 5 and 6). Leachate lead loads from these columns ranged from 0.001 to 0.02 g/day (Figure 84), corresponding to an overall decrease of 53 percent when compared to the control columns containing the untreated pyrite waste. In comparison, layering the sludge and fertilizer over the lime treated mine waste (columns 7 and 8) resulted in an overall decrease in the lead load of 58 percent compared to the control column. Lead was the only metal that showed a higher decrease in overall load when the materials were layered than when incorporated. Based on the overall low lead levels noted in the leachates, it is highly probable that this difference in overall load decrease may be more attributed to internal experimental variation as opposed to real significant differences and therefore, may not be of any consequence.

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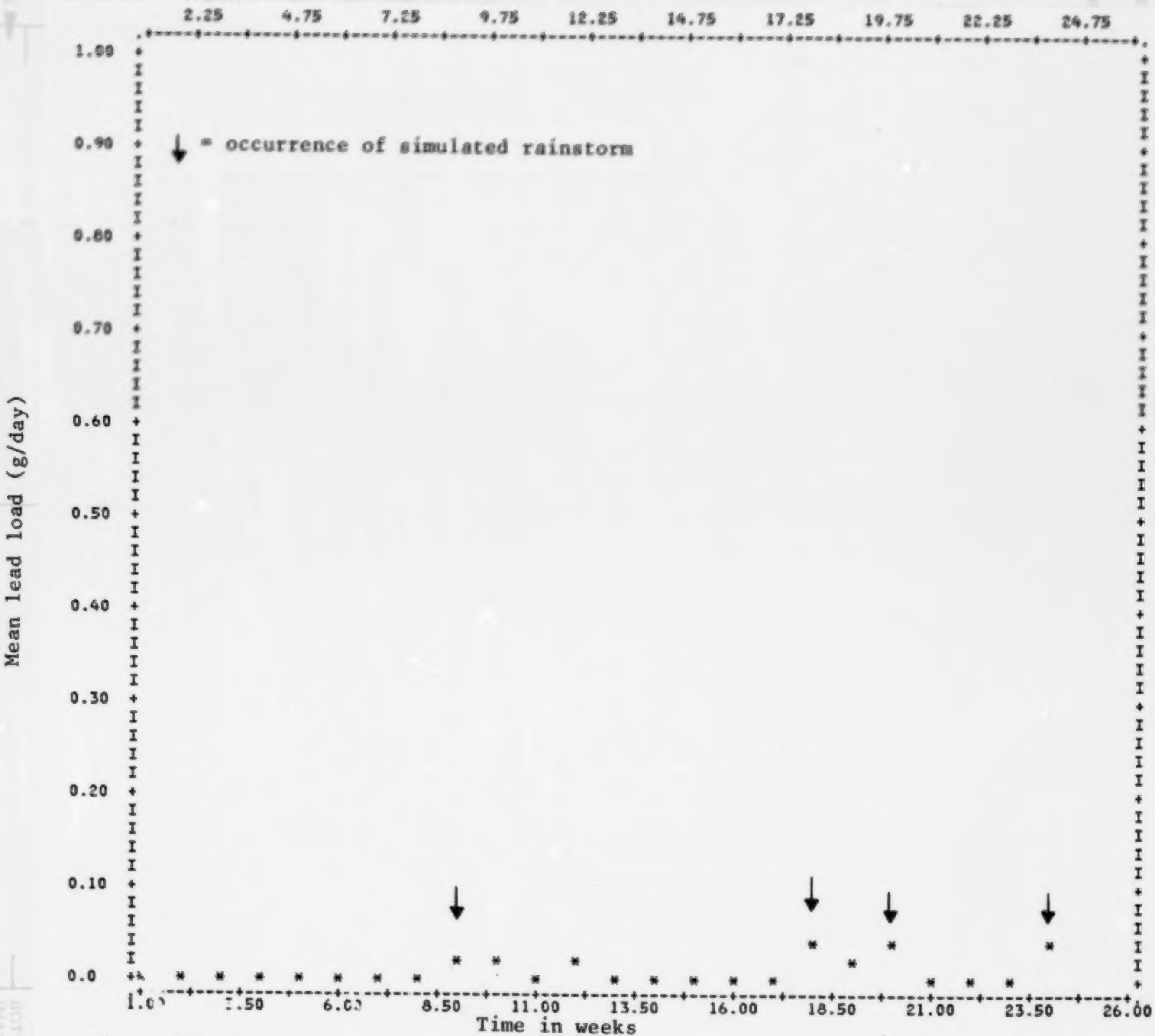

Figure 82. Graph of mean lead load as a function of time for Columns 1 and 2 (mine waste controls).

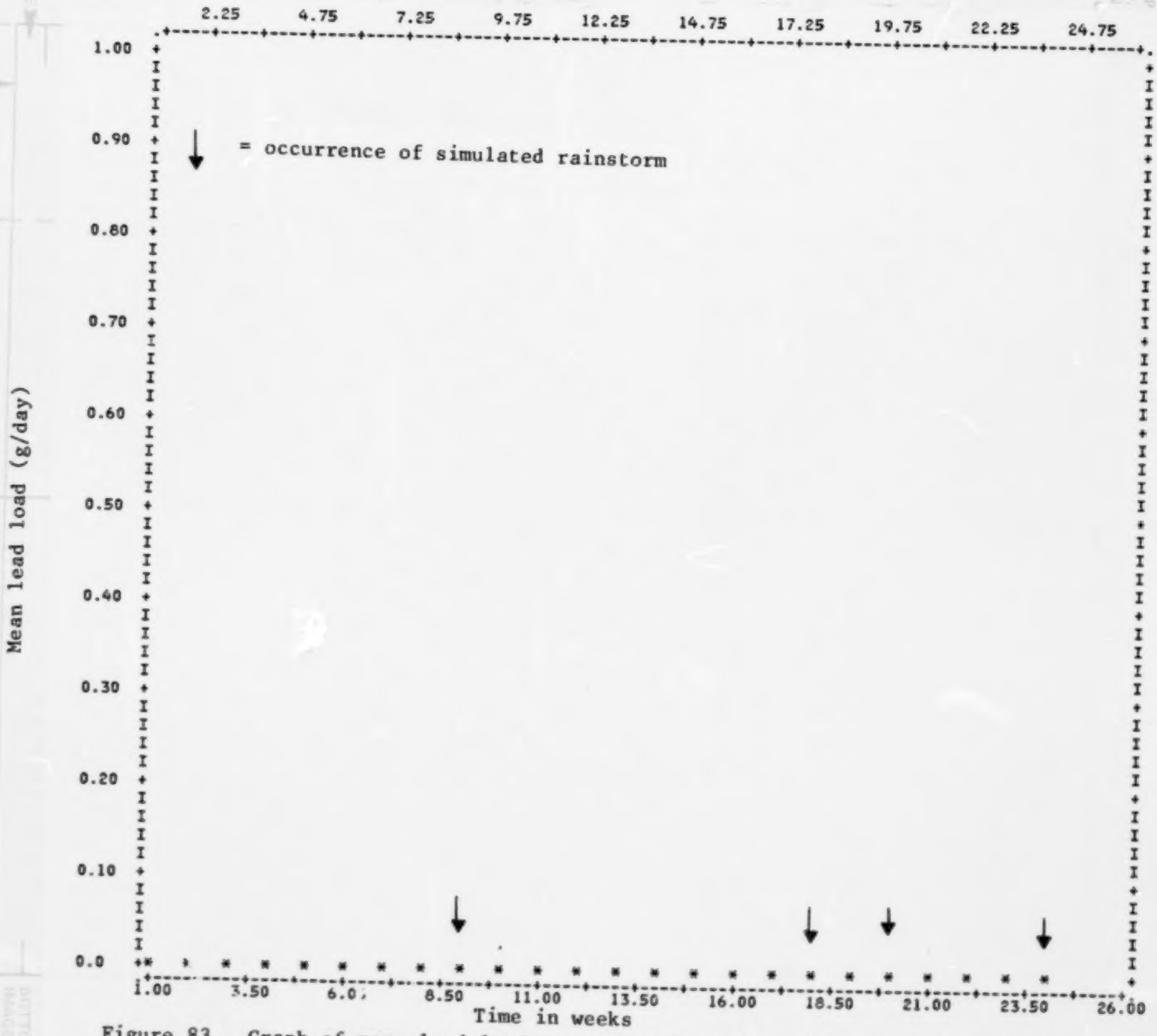

Figure 83. Graph of mean lead load as a function of time for Columns 3 and 4 (cover soil controls).

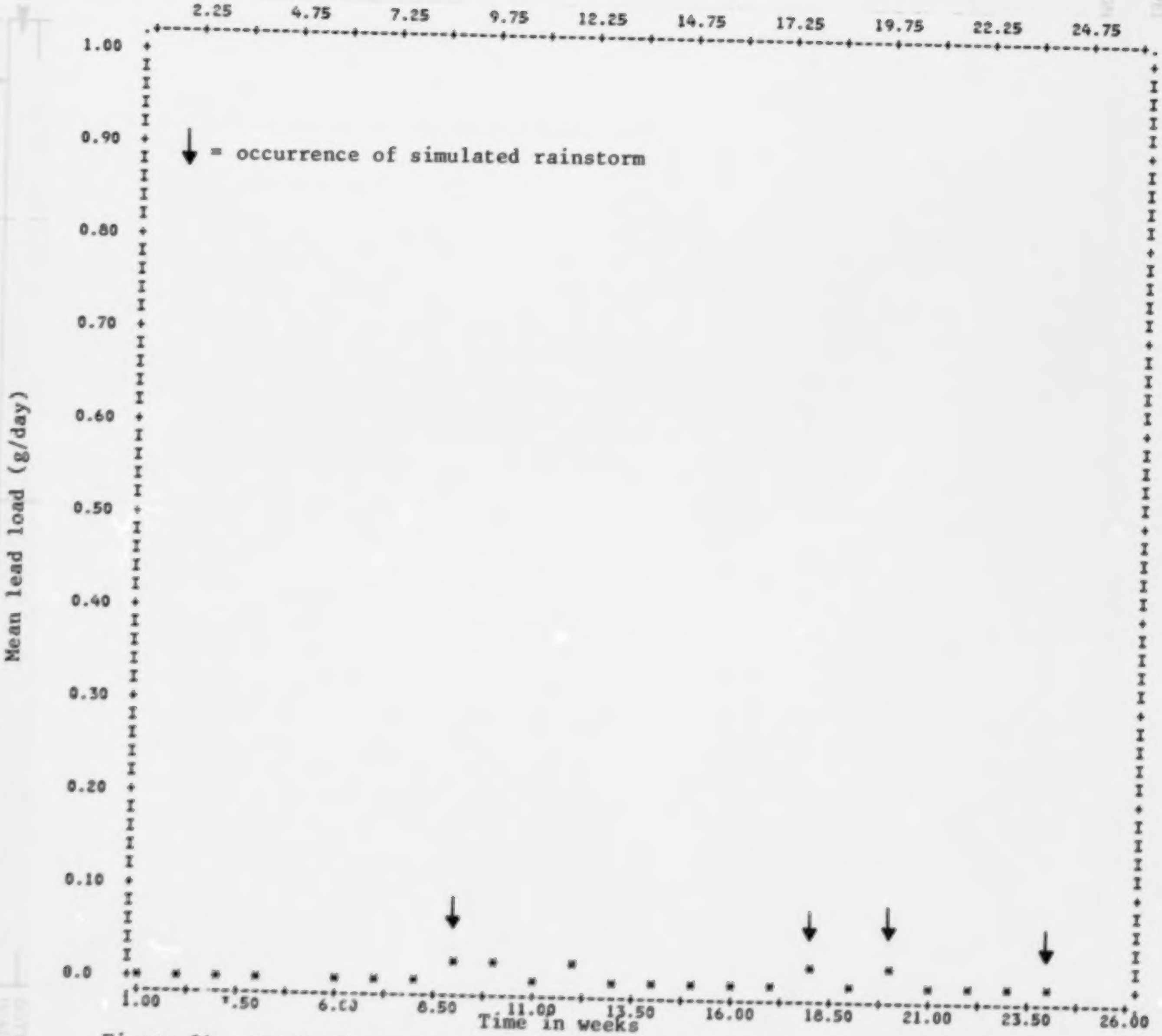

Figure 84. Graph of mean lead load as a function of time for Columns 5 and 6 (incorporated amendments).

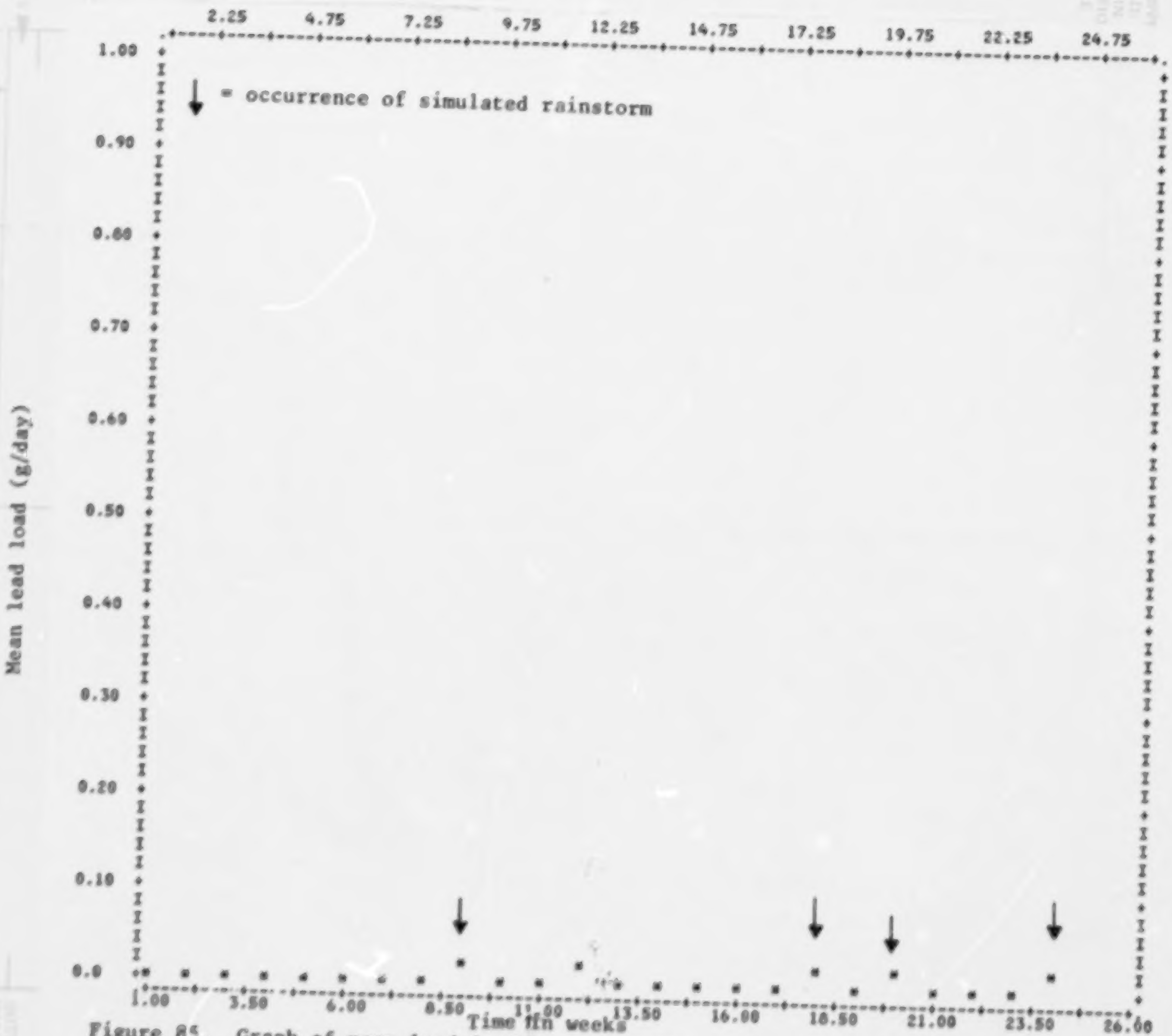

Figure 85. Graph of mean lead load as a function of time for Columns 7 and 8 (layered amendments).

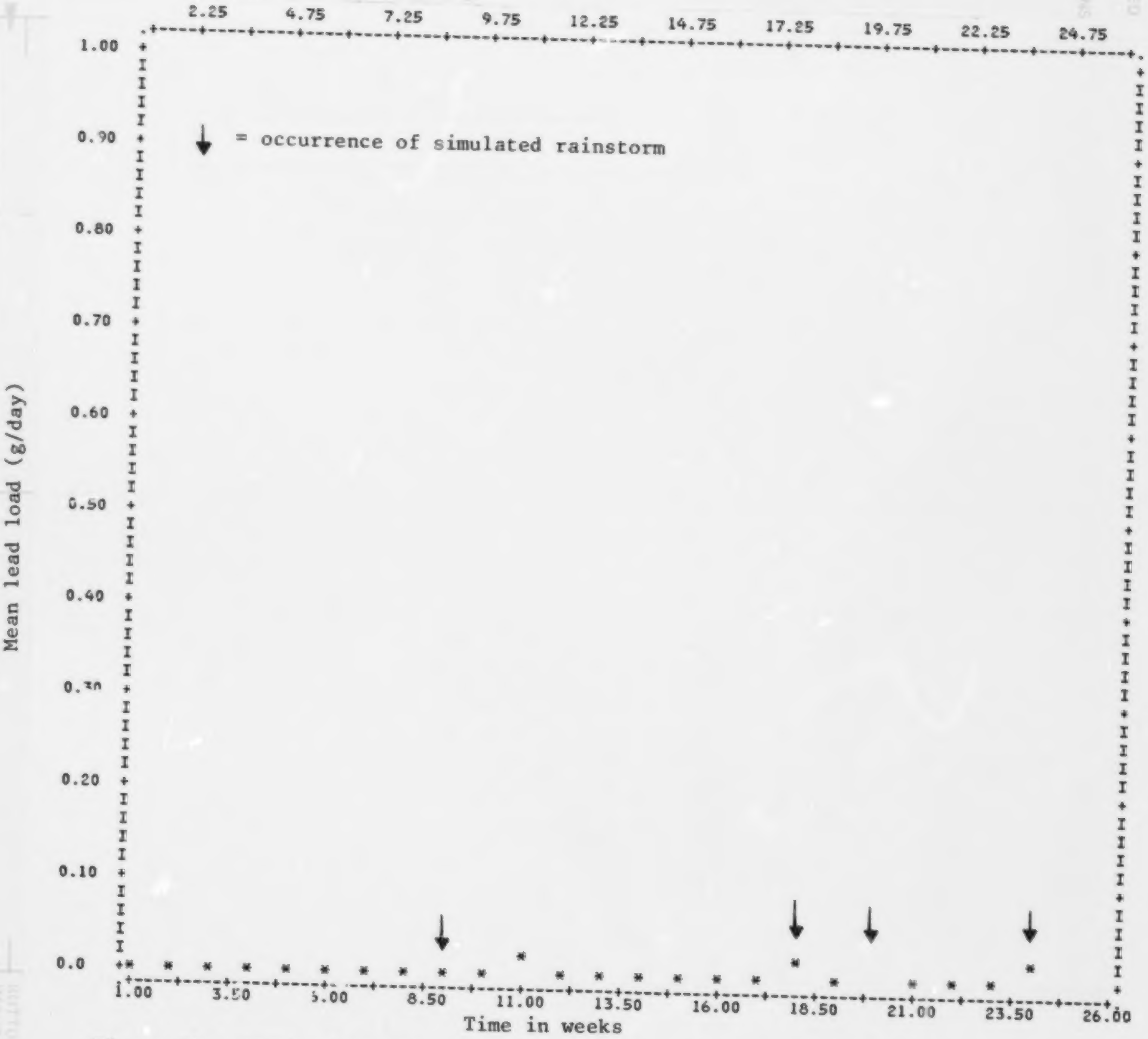

Figure 86. Graph of mean lead load as a function of time for Columns 9 and 10 (0.46-m (1.5-ft) cover soil layer).

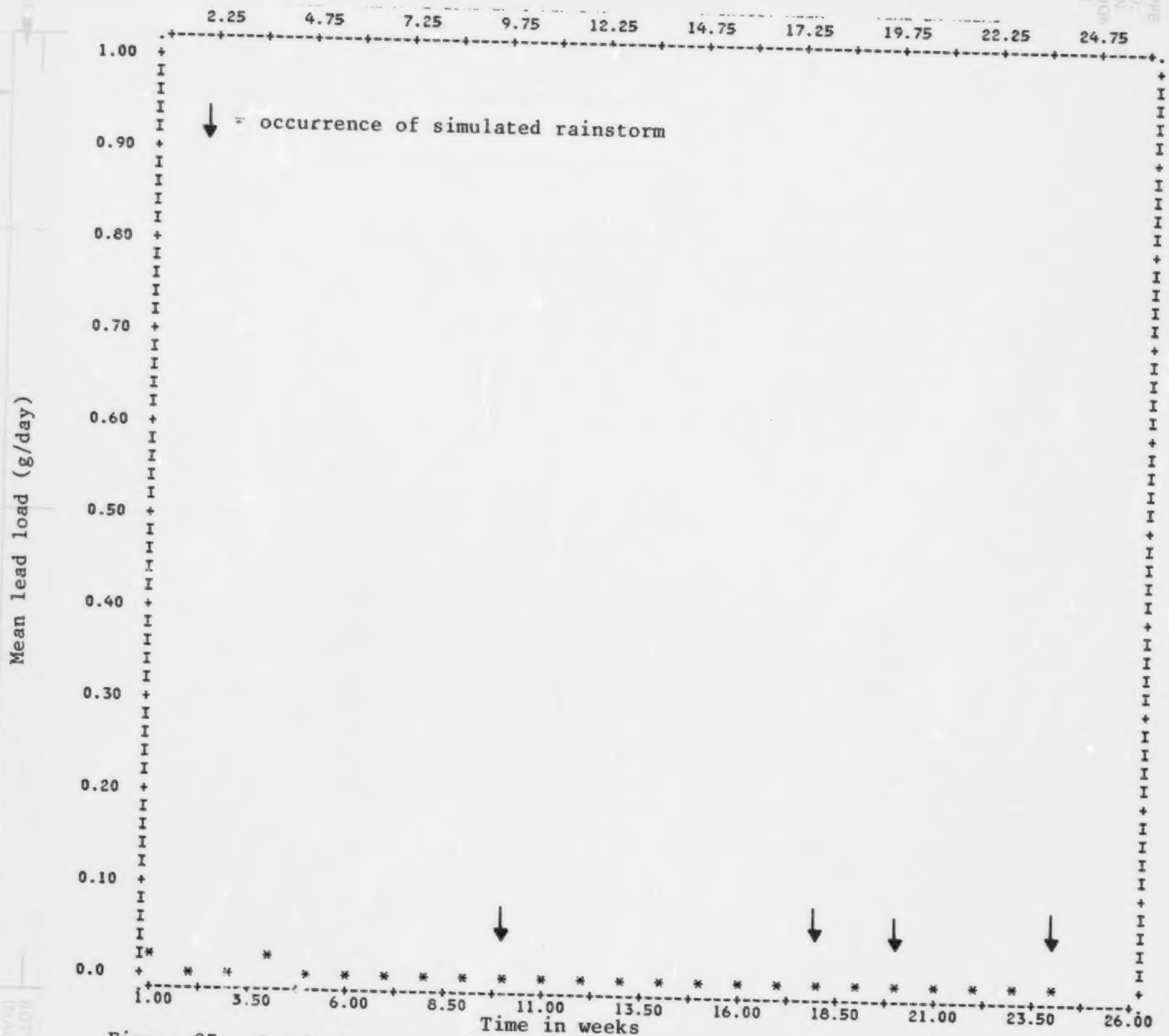

Figure 87. Graph of mean lead load as a function of time for Columns 11 and 12 (0.61-m (2-ft) cover soil layer).

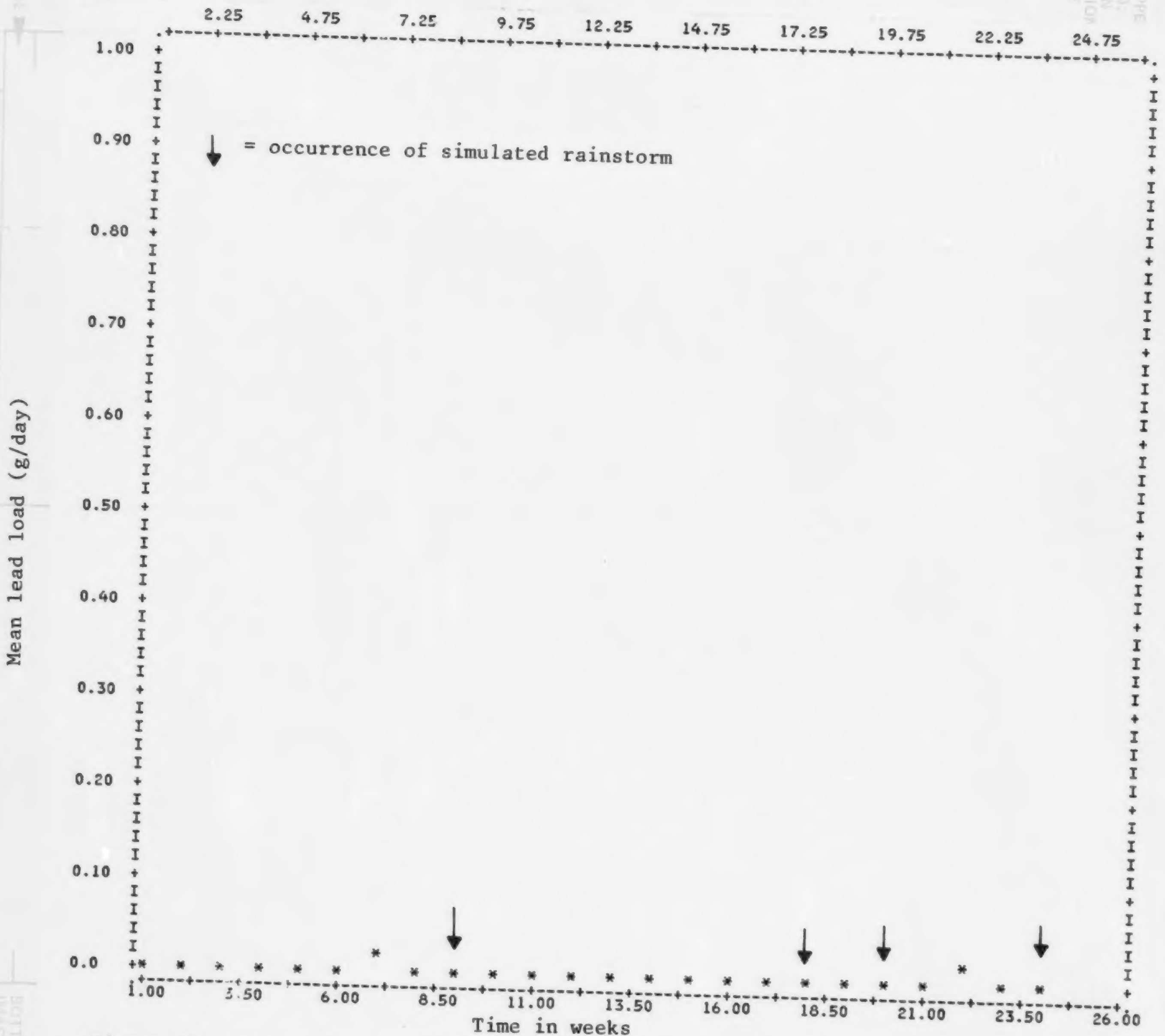

Figure 88. Graph of mean lead load as a function of time for Columns 13 and 14 (1.2-m (4-ft) cover soil layer).

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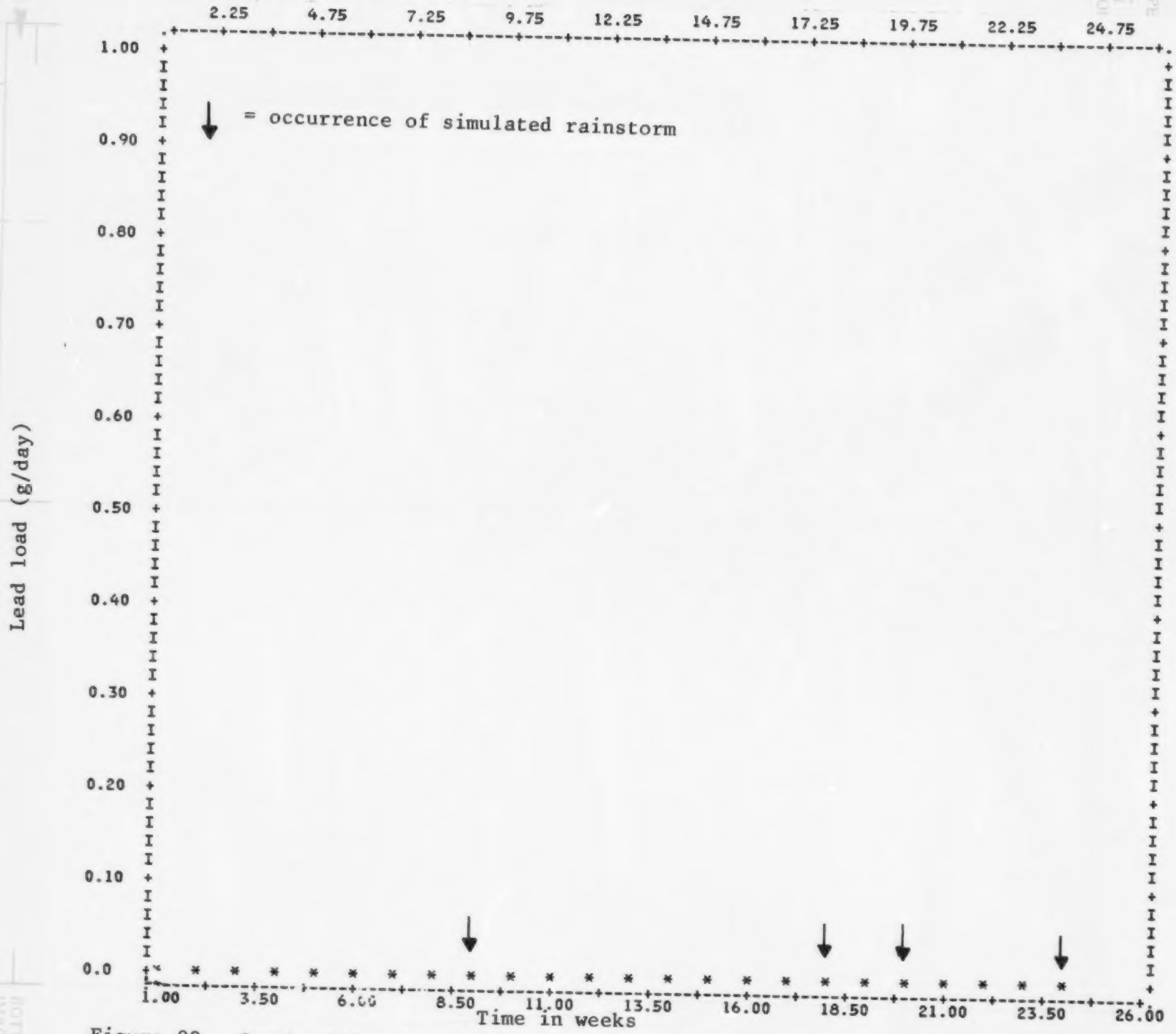

Figure 89. Graph of lead load as a function of time for Column 15 (2.5-cm (1-in) neutralizing layer).

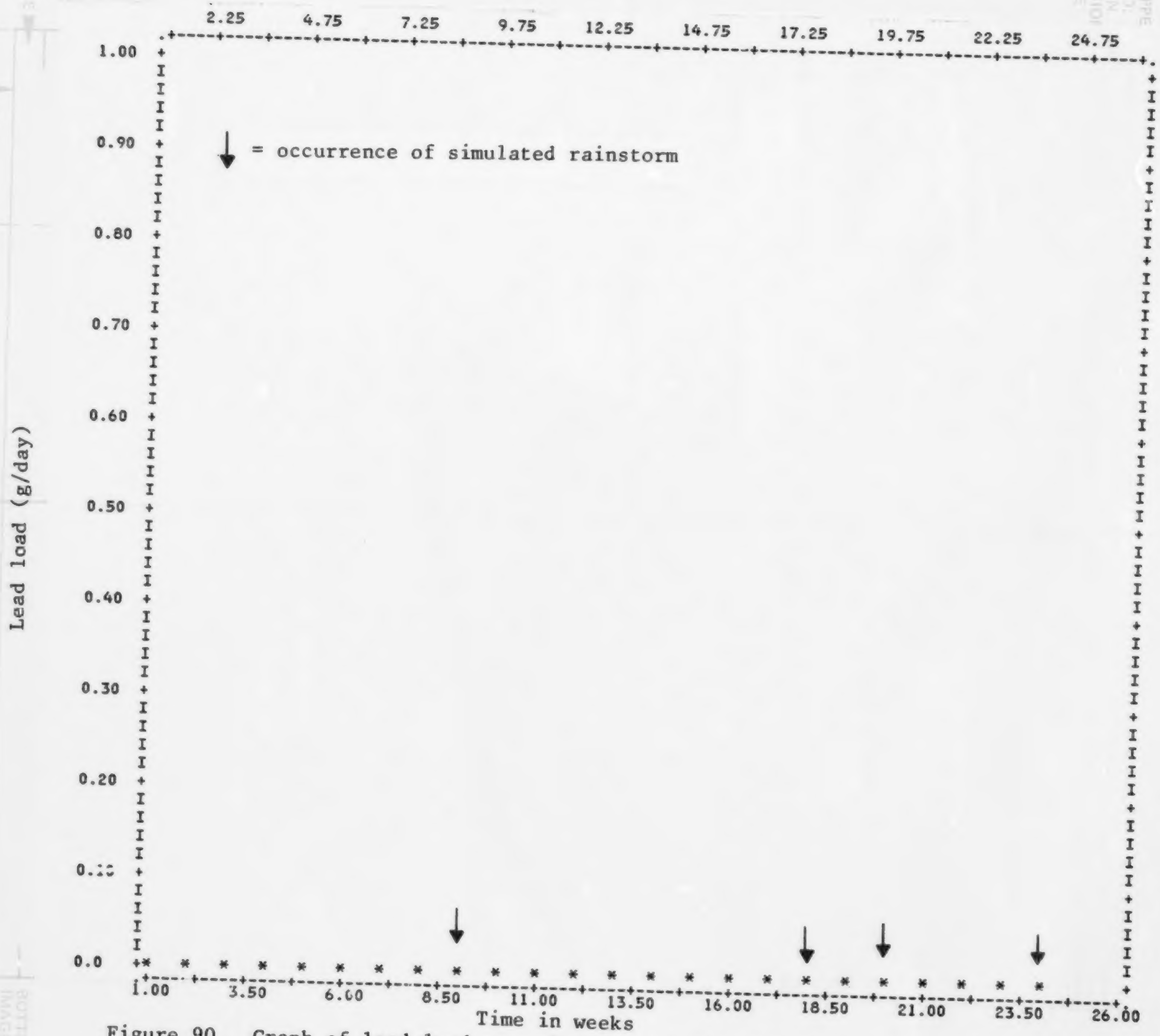

Figure 90. Graph of lead load as a function of time for Column 16 (cover soil/mine waste control).

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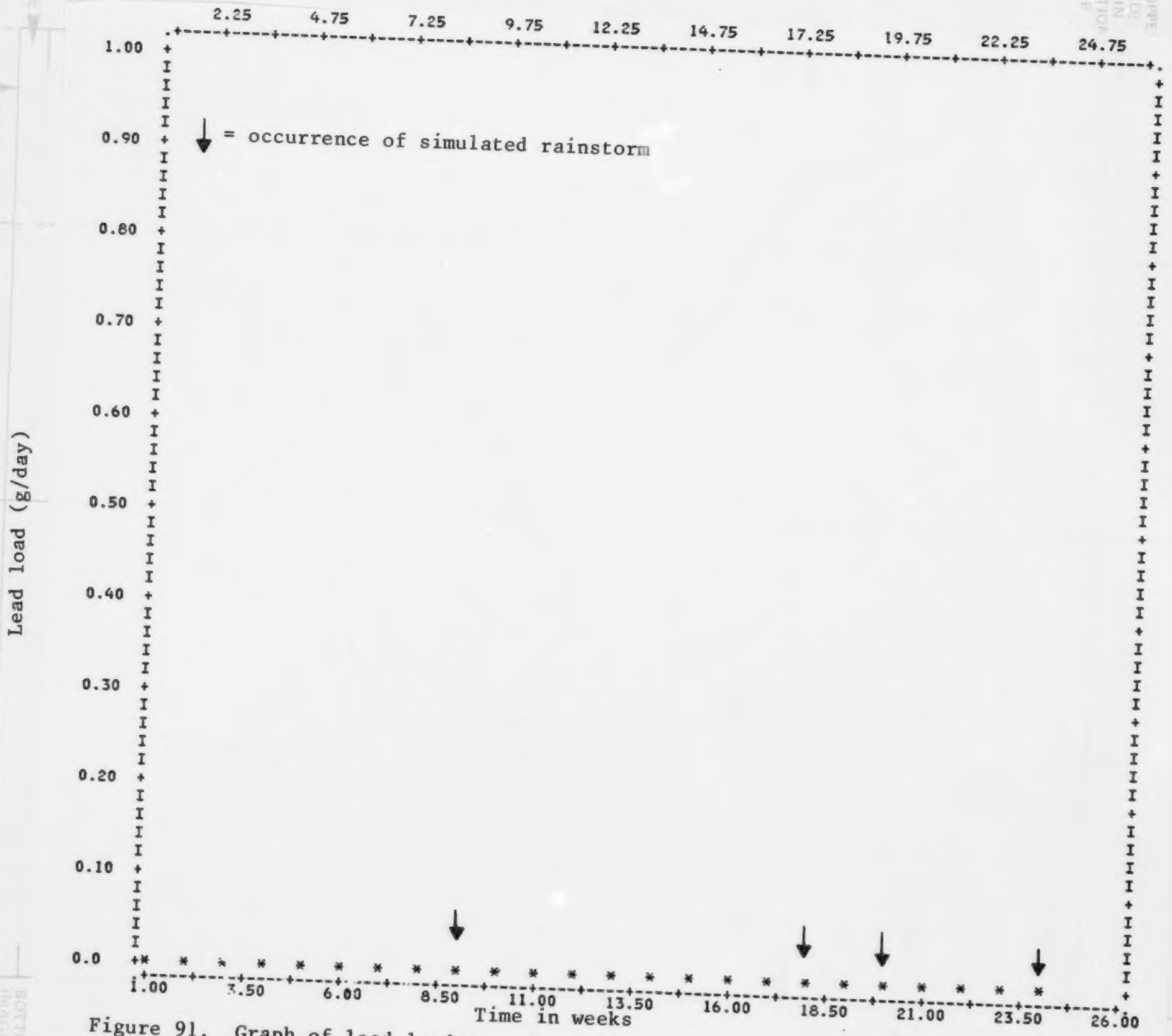

Figure 91. Graph of lead load as a function of time for Column 17 (5.0-cm (2-in) neutralizing layer).

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ASSESSMENT OF METAL MIGRATION THROUGH THE MINE WASTE AND SOIL STRATA

Overall, metal migration through the mine waste and cover soil strata was minimal during the project study period. As expected, one of the more obvious trends in these data showed that the predominant metal migration through the mine waste was in the direction of water flow. This trend appeared to be most noticeable in the mine waste control columns (1 and 2). At the end of the study period, the total iron concentration within the upper 0.31 m (1 ft) of the mine waste was approximately 1.8 to 2.4 times less than the iron concentration within the next 0.31 m to (1 ft) section of the mine waste (See Appendix E). Likewise, throughout the remaining depth of the columns, the total iron concentration in each 0.31-m (1-ft) section was approximately 1.1 times greater than the previous section. This same trend was noted with the copper migration data as well. As with iron, compounded increases with respect to depth were noted. In the mine waste control columns, the total copper concentration increased 1.0 to 1.3 times with each 0.31-m (1-ft) increase in column depth. These patterns and concentration ranges were observed in the mine waste sections of most of the experimental columns as well. Furthermore, data from metal analyses showed very little zinc and lead migration through the mine waste over the course of the study period.

In comparison, data from the metal analyses performed on the cover soil material showed no apparent downward migration patterns for any of the metals mentioned above. Metal concentrations were considerably lower in the cover soil, as expected. It did not appear that the trace metal concentrations in the cover soil or wastewater sludge contributed significantly to the metal concentrations in the mine waste. Also, no significant metal migration patterns or concentration changes were detected at any of the cover soil/mine waste or mine waste/amendment interfaces in the columns.

Overall, very small portion of the potentially leachable heavy metals in the mine waste were actually leached during this study. Rough calculations performed using the data collected from the mine waste control columns (1 and 2) showed that approximately 1% of the copper was leached from the mine waste in these columns. The following sample calculations apply to column 1:

(1)	Mass of mine waste in column (Table 6)	X	copper concentration of mine waste (Table 7)	=	Potentially leachable copper
	153 kg	X	9.47 mg/g	=	1,448,910 mg copper

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EPA Form 2350-6 (4-80)
(PREVIOUSLY EIP. EPA FORM 241)
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(2) Total volume of leachate collected from column (Appendix A) X Column leachate weekly copper concentration average (Appendix C) = Actual quantity of copper leached

82.7 liters X 223 mg/1 = 18,442 mg copper

(3) (Actual copper leached/ potentially leachable copper) X 100 = Percent of copper leached from column during study

(18,442 mg / 1,448,910 mg) X 100 = 1.3%

Similar calculations show that approximately 10% of the potentially leachable iron, 0.02% of the lead, and 1.5% of the zinc were actually leached from the mine waste in these control columns during the study. These data and calculations are rough and serve only as estimates since they do not take into account moisture content, initial fracturing, column effects, and other physical, chemical, and experimental conditions affecting metal leaching.

VEGETATIVE METAL UPTAKE DETERMINATIONS

Vegetative metal uptake was monitored intermittently during the project. By the end of the study period, vegetation was observed on all columns except those containing the untreated mine waste, which was anticipated. Vegetation was the most dense on the soil control columns (3 and 4) and the least dense on columns 7 and 8 containing the fertilizer and sludge layered over the lime-treated mine waste. Vegetative growth on the sludge incorporated columns (5 and 6) was more dense than growth on the layered sludge columns (7 and 8).

Overall, the rate of vegetative growth was much slower on the sludge treated columns (5 through 8) since no cover soil had been applied and vegetation had to be established on the surface of the treated mine waste. However, growth on these columns indicated that even the low application of digested wastewater sludge used supplied sufficient nutrients for vegetation establishment and growth. At the end of the six-month study period, the vegetation on columns 3 and 4 averaged 1.1 m (3.5 ft) in height. Vegetation on the experimental columns containing the cover soil measured close to that on the soil control columns, while vegetation on the sludge treated columns averaged 0.3 m (1 ft) in height.

Results of the vegetative analyses for trace metals indicated that many concentrations were higher in the columns containing the sludge treated mine waste (columns 5 through 8). This may be attributed to the pH of the surface material. The surface soil pH in these columns was

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(PREVIOUSLY CIN. EPA FORM 287)
TYPING GUIDE SHEET

approximately 3, whereas, in the columns containing the cover soil placed over the mine waste (columns 9 through 17), the surface soil pH was closer to 5. Vegetative uptake data as recorded at the end of the study period are tabulated in Appendix D. In general, it was noted copper uptake concentrations were predominately higher when the amendments were incorporated into the upper portion of the mine waste (columns 5 and 6) than when layered over the lime-treated mine waste (columns 7 and 8). Iron uptake, however, was greater in the latter case while lead, and zinc uptake showed little differentiation with respect to either amelioration technique (see Appendix D). Overall, the vegetation on columns containing the cover soil layered above the mine waste showed minimal metal uptake.

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DISCUSSION

The results of this research illustrate that the quality of leachate emanating from this pyrite mine waste may be notably improved by all treatments utilized in this study, i.e., incorporating lime, sludge and fertilizer into the upper strata of the mine waste, placing a mixture of sludge and fertilizer on top of the lime-treated mine waste, or by covering the mine waste with a 0.6-to 1.2-m (2-ft)to 4-ft) layer of cover soil (Table 9). In addition, by the end of the six-month study period a vegetative cover had been established on the soil surface and the exposed treated mine waste surface in all of the experimental columns. Results obtained from this research support related research studies demonstrating the successful utilization of sewage sludge in mine reclamation efforts.

A soil cover layer greater than 0.6 m (2 ft) placed over the mine waste probably represents the best overall treatment of those examined in this study. This technique not only tends to improve the quality of leachate resulting from the waste but also provides a suitable stratum for the immediate establishment of vegetation. The data generated from this study indicate that leachate quality may decrease during major rainfall events when a cover soil layer less than 0.6 m (2 ft) is used. An immediate and direct correlation between the occurance of rainstorm events and substantial increases in specific conductance, metal loads and acidity of the leachates from the columns containing a 0.46-m (1.5-ft) cover soil layer positioned over the mine waste (columns 9 and 10) was observed. In addition, it has been noted that a thinner layer of cover soil may serve as a poorer oxygen barrier and cause the vegetation to be more susceptible to drought conditions. Also, wind and water erosion during plant establishment can cause displacement of a thin soil layer and subsequent exposure of the mine waste.

In columns containing a layer of lime positioned between the cover soil and pyrite waste, acidity and metal leachate quality did not appear to improve when compared to that of leachate from the control column containing cover soil, pyrite waste and no lime layer; in fact, leachate quality tended to be worse than that of the control column. This observation may be related to the leachate flow rate. It was noted, as has been previously discussed, that the columns containing the lime layer had a 15 - 20% lower flow rate than that of the control column without the lime layer. This decreased flow rate may have allowed for a greater build-up of Fe^{+3}, SO_4^{-2}, etc. in the waste bed, which in turn may have resulted in the increase of acidity and metal levels noted in the leachates from these columns (columns 15 and 17).

These data are in agreement with previous research on acidic mine spoils where leaching solutions were allowed to pass through limestone chips prior to contacting Pennsylvania overburden materials of varying sulfur contents.

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EPA Form 2350-4 (4-80)
(PREVIOUSLY CIN. EPA FORM 287)
TYPING GUIDE SHEET

TABLE 9. EXPERIMENTAL COLUMN PERCENT LOAD DECREASES AS COMPARED TO THE UNTREATED MINE WASTE CONTROL

		Percent Decrease				
Columns	Treatment	Mean Acidity Load	Mean Copper Load	Mean Iron Load	Mean Lead Load	Mean Zinc Load
5 and 6	Incorporated lime/sludge/ fertilizer into mine waste	77	58	67	53	67
7 and 8	Fertilizer/sludge layered over lime-treated waste	65	44	55	58	51
9 and 10	0.46-m (1.5-ft) coversoil	58	56	50	75	56
11 and 12	0.6-m (2-ft) coversoil	71	54	60	79	53
13 and 14	1.2-m (4-ft) coversoil	81	64	71	75	69

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TYPING GUIDE SHEET

No difference was noted in acid production (Caruccio, 1968). In contrast, Infanger and Hood (1980) noted a great improvement in effluent quality when limestone was positioned above an acid-producing layer. Leaching experiments were performed utilizing various stacking sequences to determine the effect of the positioning of an acid producing overburden on effluent quality.

The lime layer may be beneficial in that it may assist in protecting the quality of the cover soil and facilitate plant growth. According to Grim and Hill (1974), soils in intimate contact with toxic spoil may become contaminated and ultimately become phytotoxic. In columns containing a lime layer (columns 15 and 17) it was noted that plant root growth actually penetrated the lime layer and extended down into the upper portion of the waste material. In the control column without the lime layer (column 16) the root development stopped above the soil/mine waste interface. Even though the roots initially grew close to this interface, it could be possible that with time, upward migration of toxic materials from the waste would eventually toxify the soil immediately above the interface so that root development in this zone would be inhibited. At any rate, the lime layer appeared to enhance root development by extending the root zone past the cover soil into the waste material. In actual field situations application of lime to pyritic mine wastes prior to covering with top soil could be key in establishing acceptable vegetative growth particularly where thin (i.e., less than 0.6-m (2-ft)) layers of cover soil are used. In these situations the lime layer could assist in protecting the quality of the cover soil while providing an extended root zone.

Layering and/or mixing the upper portion of pyritic wastes with lime, sewage sludge, and fertilizer may offer a realistic solution in areas where limited quantities of cover soil are available. The results of this six-month study indicate that the application of these materials may substantially improve leachate quality, with the performance noted comparable to the 1.2-m (4-ft) cover soil treatment.

The data from this experiment indicate that incorporation of the sludge, lime, and fertilizer into the upper strata of the mine waste may be the more beneficial of the two sludge/lime application techniques (Figures 92 to 97). There appeared to be greater control of metals and acidity in the leachate during simulated rainstorm events. This difference may have been due to the mixing process inherently increasing the surface to volume ratio of the amendment particles and more evenly distributing them through the upper column layer so that in effect there were more "available" amendments in the mixed columns than in the layered columns.

Regardless of the technique (i.e., incorporation or layering) utilized, treatment with sewage sludge appears to offer attractive efficacy as well as economic advantages particularly since in some cases sewage sludge can be obtained free or at a very low cost. In this case, the costs incurred in the sludge layering treatment may be lower than those associated with the hauling and spreading of cover soil.

A disadvantage of both the sludge incorporation and layering treatment techniques is the very thin layer of treated material. The root penetration

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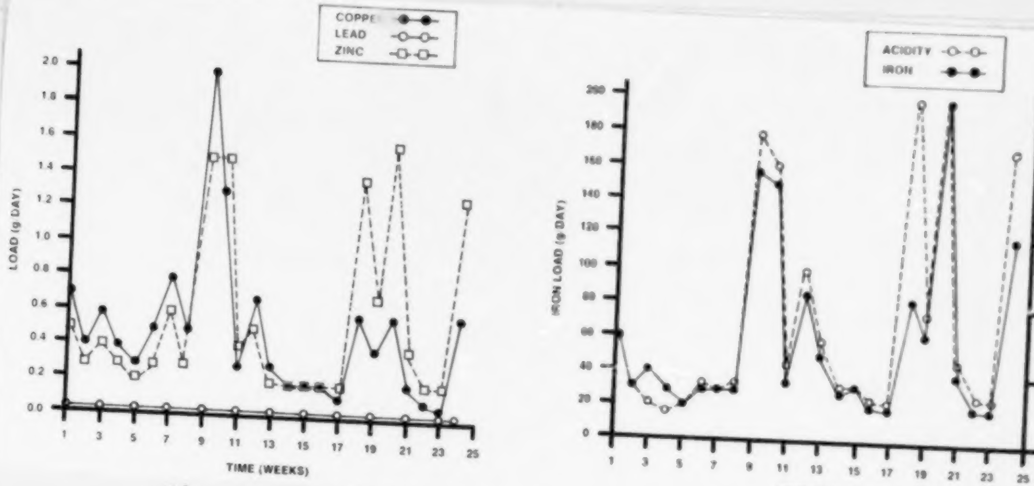

Figure 92: Combined Metal and Acidity Mean Load Data for Columns 1 and 2 Containing the Untreated Mine Waste.

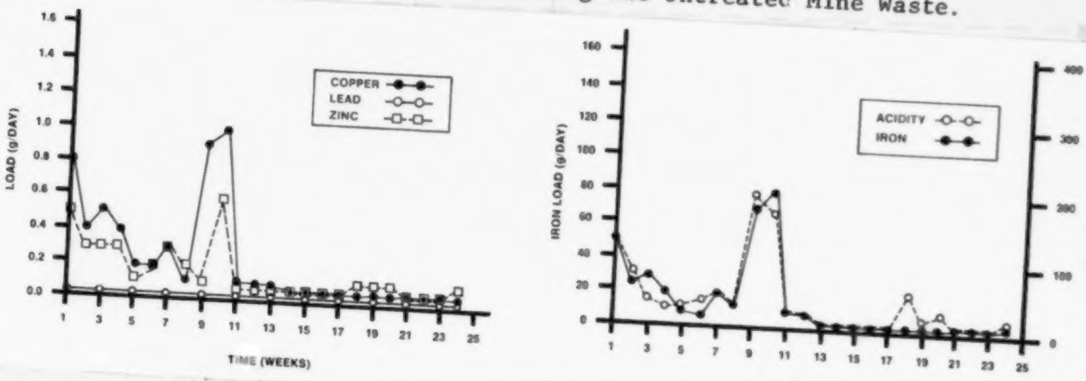

Figure 93: Combined Metal and Acidity Mean Load Data for Columns 5 and 6 Containing the Lime/Sludge/Fertilizer Layered over the Lime-Treated Mine Waste.

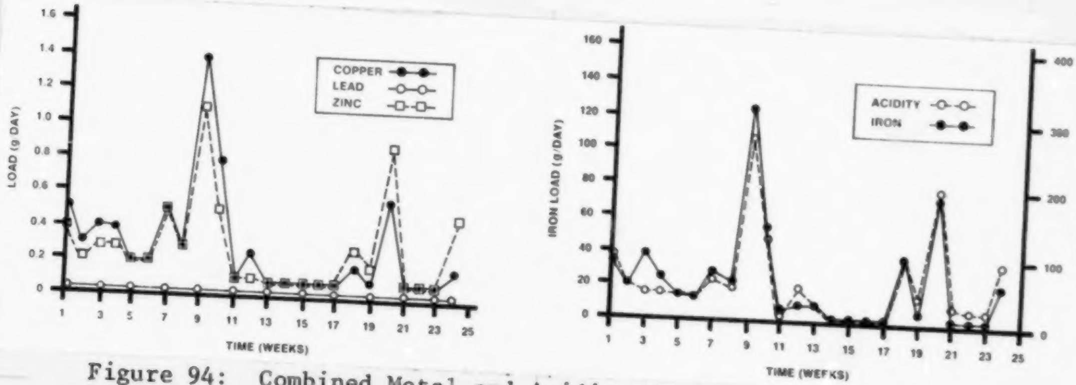

Figure 94: Combined Metal and Acidity Mean Load Data for Columns 7 and 8 Containing a Sludge/Fertilizer Layer over the Lime-Treated Mine Waste.

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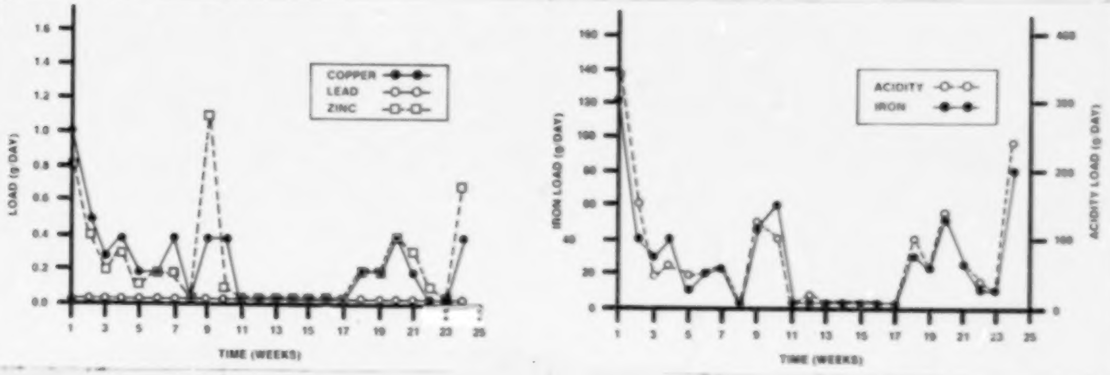

Figure 95: Combined Metal and Acidity Load Data for Columns 9 and 10 Containing the Mine Waste under 0.46m (1.5ft) of Cover Soil.

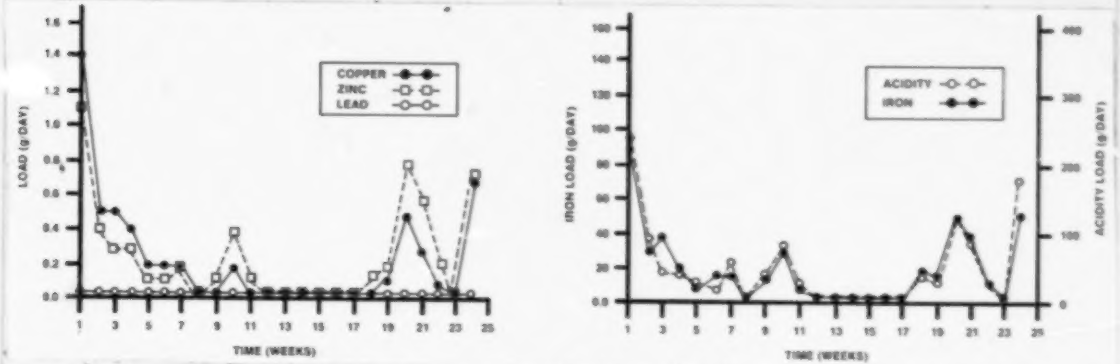

Figure 96: Combined Metal and Acidity Load Data for Columns 11 and 12 Containing the Mine Waste under 0.61m (2 ft) of Cover Soil.

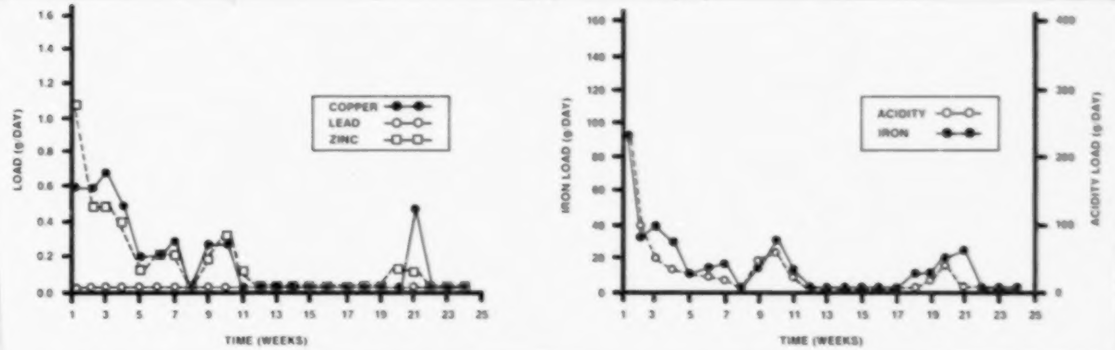

Figure 97: Combined Metal and Acidity Load Data for Columns 13 and 14 Containing the Mine Waste under 1.2m (4 ft) of Cover Soil.

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TYPING GUIDE SHEET

of the vegetation is usually very shallow and the vegetation may be subject to adverse effects under drought conditions. This problem has been noted at the Contrary Creek Demonstration project site. It is also possible that this effect may be more acute with the layering technique since it was observed during this study that the growth rate of vegetation on columns 5 and 6 (incorporated materials) was faster and the vegetation more dense when compared to columns 7 and 8 (layered configurations).

Nevertheless, the data indicate that in lieu of covering pyritic mine wastes with a heavy layer of cover soil, treating the top layer of the waste with municipal sludge, fertilizer, and lime may offer the next best available option. The validity of this viewpoint is supported by the field data generated at Contrary Creek (Hinkle, 1980) and Tioga River (Miorin _et_ _al_., 1979; Hill _et_ _al_., 1977).

The storm events simulated in these studies resulted in peak loads of leachate acidity and metals (Figure 92 to 97). These results are in keeping with the field observations of Nordstrom and Dagenhart (1979) who found that storm events led to high concentrations of acidity and metals in the waters of Contrary Creek. This was attributed to heavy rainfalls flushing pyritic oxidation products from the waste piles and into the creek. These oxidation products were believed to build up in the pyritic wastes during relatively dry periods between storms so that the waste piles served as a reservoir for these products.

In the greenhouse column studies, there was considerable variation in the storm actuated leachate acidity and metal peaks. In columns containing the mine waste covered with 1.2 m (4 ft) of cover soil, these peaks were considerably lessened. Presumably, this was due at least in part to the heavy layer of soil acting as an oxygen barrier and consequently retarding the oxidation of the pyritic wastes with less subsequent formation of acid, etc. Columns containing less cover soil (i.e., 0.61 m and 0.46 m) showed much higher acidity and metal peaks following rain storms than did the columns containing 1.2 m (4 ft) of cover soil.

Next to covering pyritic wastes with 1.2-m (4-ft) of cover soil, mixing sludge, lime, and fertilizer into the top layer seems to lead to the best leachate quality following storm events; layering of these components is not nearly as effective.

Columns 5 and 6 simulated the reclamation techniques used at the Contrary Creek site where the average cost of reclamation, including all maintenance for two of the mine sites, was $14,170 per hectare ($5,735/acre) (Hinkle, 1980).

In summary covering the waste with 1.2 m (4 ft) of cover soil not only presents the best option for improving overall leachate quality in the field, but for maintaining leachate quality particularly during storms when the highest acid and metal loads would be anticipated. If this option is not possible, mixing sludge, fertilizer, and lime into the upper layer of the waste appears to be the next best technique.

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APPENDIX A

COLUMN LEACHATE FLOW DATA

EXPLANATION OF DATA:

COL NO. = Column number ranging from 1 to 17.

WEEK NO. = Week of study period, ranging from 1 to 24.

DIST-WATER APPLIED (LITERS) = The quantity of distilled water applied weekly to each column to simulate precipitation.

RETENTION TIME, (HRS) = The length of time from when water was applied to the columns until leachates were collected for analysis.

LEACHATE COLLECTED, (LITERS) = The quantity of leachate collected.

LEACHATE FLOW RATE, LITERS/HOUR = The calculated rate of leachate flow from a column.

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APPENDIX A (continued).

WEEK NO.	COL NO.	DIST-WATER APPLIED (LITERS)	RETENTION TIME (HRS)	LEACHATE COLLECTED (LITERS)	LEACHATE FLOW RATE (LITERS/HR)
1.	1.	1.5	23.50	2.5	0.106
1.	2.	1.5	23.50	2.0	0.085
1.	3.	1.5	23.50	2.0	0.085
1.	4.	1.5	23.50	2.3	0.098
1.	5.	1.5	23.50	2.7	0.115
1.	6.	1.5	23.50	1.5	0.064
1.	7.	1.5	23.50	1.3	0.055
1.	8.	1.5	23.50	2.0	0.085
1.	9.	1.5	23.50	3.0	0.128
1.	10	1.5	23.50	4.0	0.170
1.	11	1.5	23.50	4.3	0.183
1.	12	1.5	23.50	4.2	0.179
1.	13	1.5	23.50	5.0	0.213
1.	14	1.5	23.50	3.6	0.153
1.	15	1.5	23.50	3.0	0.128
1.	16	1.5	23.50	2.0	0.085
1.	17	1.5	23.50	2.3	0.098
2.	1.	1.5	23.50	1.8	0.077
2.	2.	1.5	23.50	1.0	0.043
2.	3.	1.5	23.50	1.0	0.043
2.	4.	1.5	23.50	1.2	0.051
2.	5.	1.5	23.50	1.3	0.055
2.	6.	1.5	23.50	1.3	0.055
2.	7.	1.5	23.50	1.0	0.043
2.	8.	1.5	23.50	1.0	0.043
2.	9.	1.5	23.50	2.1	0.089
2.	10	1.5	23.50	2.0	0.085
2.	11	1.5	23.50	2.0	0.085
2.	12	1.5	23.50	2.0	0.085
2.	13	1.5	23.50	2.3	0.098
2.	14	1.5	23.50	2.1	0.089
2.	15	1.5	23.50	2.0	0.085
2.	16	1.5	23.50	1.2	0.051
2.	17	1.5	23.50	1.5	0.064
3.	1.	1.5	26.00	1.3	0.050
3.	2.	1.5	26.00	0.8	0.031
3.	3.	1.5	26.00	0.9	0.035
3.	4.	1.5	26.00	1.4	0.038
3.	5.	1.5	26.00	0.8	0.031
3.	6.	1.5	26.00	0.8	0.031
3.	7.	1.5	26.00	0.8	0.031
3.	8.	1.5	26.00	1.0	0.038
3.	9.	1.5	26.00	0.6	0.023
3.	10	1.5	26.00	0.9	0.035
3.	11	1.5	26.00	1.1	0.042
3.	12	1.5	26.00	1.0	0.038
3.	13	1.5	26.00	1.3	0.050
3.	14	1.5	26.00	1.5	0.058
3.	15	1.5	26.00	1.0	0.038
3.	16	1.5	26.00	0.8	0.031
3.	17	1.5	26.00	1.0	0.038
4.	1.	1.5	25.00	0.9	0.036
4.	2.	1.5	25.00	0.9	0.036
4.	3.	1.5	25.00	0.5	0.020
4.	4.	1.5	25.00	0.8	0.032
4.	5.	1.5	25.00	0.8	0.032
4.	6.	1.5	25.00	0.6	0.024
4.	7.	1.5	25.00	0.8	0.032
4.	8.	1.5	25.00	0.9	0.036
4.	9.	1.5	25.00	1.0	0.040
4.	10	1.5	25.00	1.0	0.040
4.	11	1.5	25.00	1.0	0.040
4.	12	1.5	25.00	0.8	0.032
4.	13	1.5	25.00	1.1	0.044
4.	14	1.5	25.00	1.3	0.052
4.	15	1.5	25.00	0.9	0.036
4.	16	1.5	25.00	0.4	0.016
4.	17	1.5	25.00	0.9	0.036
5.	1.	2.5	25.75	1.3	0.050
5.	2.	2.5	25.75	1.0	0.039
5.	3.	2.5	25.75	0.7	0.027
5.	4.	2.5	25.75	1.0	0.039
5.	5.	2.5	25.75	0.6	0.023
5.	6.	2.5	25.75	0.8	0.031
5.	7.	2.5	25.75	1.0	0.039
5.	8.	2.5	25.75	1.0	0.039
5.	9.	2.5	25.75	0.9	0.035
5.	10	2.5	25.75	0.8	0.031
5.	11	2.5	25.75	0.9	0.035
5.	12	2.5	25.75	0.7	0.027
5.	13	2.5	25.75	0.8	0.031
5.	14	2.5	25.75	1.0	0.039
5.	15	2.5	25.75	0.9	0.035
5.	16	2.5	25.75	0.2	0.008
5.	17	2.5	25.75	0.9	0.035
6.	1.	2.5	26.00	2.0	0.077
6.	2.	2.5	26.00	1.4	0.054
6.	3.	2.5	26.00	0.8	0.031
6.	4.	2.5	26.00	0.8	0.031
6.	5.	2.5	26.00	1.0	0.038
6.	6.	2.5	26.00	1.0	0.038
6.	7.	2.5	26.00	0.9	0.035
6.	8.	2.5	26.00	1.0	0.038
6.	9.	2.5	26.00	1.0	0.038
6.	10	2.5	26.00	0.9	0.035
6.	11	2.5	26.00	1.0	0.038
6.	12	2.5	26.00	0.8	0.031
6.	13	2.5	26.00	0.9	0.035
6.	14	2.5	26.00	1.0	0.038
6.	15	2.5	26.00	0.6	0.023
6.	16	2.5	26.00	0.6	0.023
6.	17	2.5	26.00	0.9	0.035

APPENDIX A (continued).

WEEK NO.	COL NO.	DIST-WATER APPLIED (LITERS)	RETENTION TIME (HRS)	LEACHATE COLLECTED (LITERS)	LEACHATE FLOW RATE (LITERS/HR)	WEEK NO.	COL NO.	DIST-WATER APPLIED (LITERS)	RETENTION TIME (HRS)	LEACHATE COLLECTED (LITERS)	LEACHATE FLOW RATE (LITERS/HR)
7.	1.	2.5	23.50	1.9	0.081	10.	1.	2.2	28.75	6.0	0.209
7.	2.	2.5	23.50	1.2	0.051	10.	2.	2.2	28.75	5.5	0.191
7.	3.	2.5	23.50	0.4	0.017	10.	3.	2.2	28.75	2.4	0.083
7.	4.	2.5	23.50	0.5	0.021	10.	4.	2.2	28.75	2.7	0.094
7.	5.	2.5	23.50	1.3	0.055	10.	5.	2.2	28.75	6.2	0.216
7.	6.	2.5	23.50	0.9	0.038	10.	6.	2.2	28.75	7.9	0.275
7.	7.	2.5	23.50	1.0	0.043	10.	7.	2.2	28.75	4.9	0.170
7.	8.	2.5	23.50	1.8	0.077	10.	8.	2.2	28.75	5.3	0.184
7.	9.	2.5	23.50	0.7	0.030	10.	9.	2.2	28.75	3.0	0.184
7.	10	2.5	23.50	0.8	0.034	10.	10	2.2	28.75	3.3	0.115
7.	11	2.5	23.50	0.9	0.038	10.	11	2.2	28.75	3.0	0.104
7.	12	2.5	23.50	0.6	0.026	10.	12	2.2	28.75	3.2	0.111
7.	13	2.5	23.50	1.0	0.043	10.	13	2.2	28.75	5.1	0.177
7.	14	2.5	23.50	0.7	0.030	10.	14	2.2	28.75	4.3	0.150
7.	15	2.5	23.50	0.1	.004	10.	15	2.2	28.75	2.0	0.070
7.	16	2.5	23.50	0.1	.004	10.	16	2.2	28.75	2.0	0.070
7.	17	2.5	23.50	0.8	0.034	10.	17	2.2	28.75	2.9	0.101
8.	1.	2.5	23.75	2.0	0.084	11.	1.	2.2	25.00	1.5	0.060
8.	2.	2.5	23.75	1.8	0.076	11.	2.	2.2	25.00	1.0	0.040
8.	3.	2.5	23.75	0.0	0.000	11.	3.	2.2	25.00	0.2	0.008
8.	4.	2.5	23.75	0.5	0.021	11.	4.	2.2	25.00	0.3	0.012
8.	5.	2.5	23.75	1.0	0.042	11.	5.	2.2	25.00	1.0	0.040
8.	6.	2.5	23.75	0.7	0.029	11.	6.	2.2	25.00	0.8	0.032
8.	7.	2.5	23.75	1.3	0.055	11.	7.	2.2	25.00	0.9	0.036
8.	8.	2.5	23.75	1.5	0.063	11.	8.	2.2	25.00	1.2	0.048
8.	9.	2.5	23.75	0.3	0.013	11.	9.	2.2	25.00	0.3	0.012
8.	10	2.5	23.75	0.3	0.013	11.	10	2.2	25.00	0.0	0.000
8.	11	2.5	23.75	0.3	0.013	11.	11	2.2	25.00	0.9	0.036
8.	12	2.5	23.75	0.3	0.013	11.	12	2.2	25.00	0.4	0.016
8.	13	2.5	23.75	0.8	0.034	11.	13	2.2	25.00	1.2	0.048
8.	14	2.5	23.75	0.5	0.021	11.	14	2.2	25.00	2.0	0.080
8.	15	2.5	23.75	0.3	0.013	11.	15	2.2	25.00	0.0	0.000
8.	16	2.5	23.75	0.3	0.013	11.	16	2.2	25.00	0.1	0.004
8.	17	2.5	23.75	0.5	0.021	11.	17	2.2	25.00	0.1	0.004
9.	1.	13.5	24.25	9.3	0.384	12.	1.	3.8	23.75	3.0	0.126
9.	2.	13.5	24.25	8.9	0.367	12.	2.	3.8	23.75	3.0	0.126
9.	3.	13.5	24.25	4.8	0.198	12.	3.	3.8	23.75	0.1	0.004
9.	4.	13.5	24.25	4.0	0.165	12.	4.	3.8	23.75	0.0	0.000
9.	5.	13.5	24.25	7.0	0.289	12.	5.	3.8	23.75	2.0	0.084
9.	6.	13.5	24.25	3.9	0.161	12.	6.	3.8	23.75	1.0	0.042
9.	7.	13.5	24.25	8.1	0.334	12.	7.	3.8	23.75	2.1	0.088
9.	8.	13.5	24.25	8.3	0.342	12.	8.	3.8	23.75	2.9	0.122
9.	9.	13.5	24.25	3.1	0.128	12.	9.	3.8	23.75	0.8	0.034
9.	10	13.5	24.25	3.1	0.128	12.	10	3.8	23.75	0.2	.008
9.	11	13.5	24.25	0.6	0.025	12.	11	3.8	23.75	0.1	0.004
9.	12	13.5	24.25	2.4	0.099	12.	12	3.8	23.75	0.1	0.004
9.	13	13.5	24.25	1.4	0.058	12.	13	3.8	23.75	0.2	0.008
9.	14	13.5	24.25	2.1	0.087	12.	14	3.8	23.75	0.9	0.038
9.	15	13.5	24.25	3.8	0.157	12.	15	3.8	23.75	0.1	0.004
9.	16	13.5	24.25	5.3	0.219	12.	16	3.8	23.75	0.0	0.000
9.	17	13.5	24.25	5.8	0.239	12.	17	3.8	23.75	0.0	0.000

APPENDIX A (continued).

WEEK NO.	COL NO.	DIST-WATER APPLIED (LITERS)	RETENTION TIME (HRS)	LEACHATE COLLECTED (LITERS)	LEACHATE FLOW RATE (LITERS/HR)	WEEK NO.	COL NO.	DIST-WATER APPLIED (LITERS)	RETENTION TIME (HRS)	LEACHATE COLLECTED (LITERS)	LEACHATE FLOW RATE (LITERS/HR)
13.	1.	1.9	22.25	1.9	0.085	16.	1.	1.9	24.25	1.5	0.062
13.	2.	1.9	22.25	1.8	0.081	16.	2.	1.9	24.25	0.8	0.033
13.	3.	1.9	22.25	0.0	0.000	16.	3.	1.9	24.25	0.1	0.004
13.	4.	1.9	22.25	0.0	0.000	16.	4.	1.9	24.25	0.1	0.004
13.	5.	1.9	22.25	1.2	0.054	16.	5.	1.9	24.25	0.4	0.016
13.	6.	1.9	22.25	0.2	0.009	16.	6.	1.9	24.25	0.3	0.012
13.	7.	1.9	22.25	1.9	0.085	16.	7.	1.9	24.25	0.4	0.016
13.	8.	1.9	22.25	1.0	0.045	16.	8.	1.9	24.25	0.4	0.016
13.	9.	1.9	22.25	0.3	0.013	16.	9.	1.9	24.25	0.3	0.012
13.	10	1.9	22.25	0.3	0.013	16.	10	1.9	24.25	0.2	0.008
13.	11	1.9	22.25	0.3	0.013	16.	11	1.9	24.25	0.1	0.004
13.	12	1.9	22.25	0.3	0.013	16.	12	1.9	24.25	0.1	0.004
13.	13	1.9	22.25	0.3	0.013	16.	13	1.9	24.25	0.2	0.008
13.	14	1.9	22.25	0.9	0.040	16.	14	1.9	24.25	0.5	0.021
13.	15	1.9	22.25	0.0	0.000	16.	15	1.9	24.25	0.0	0.000
13.	16	1.9	22.25	0.0	0.000	16.	16	1.9	24.25	0.0	0.000
13.	17	1.9	22.25	0.0	0.000	16.	17	1.9	24.25	0.0	0.000
14.	1.	1.9	23.25	1.2	0.052	17.	1.	2.2	23.50	1.4	0.060
14.	2.	1.9	23.25	0.9	0.039	17.	2.	2.2	23.50	0.8	0.034
14.	3.	1.9	23.25	0.0	0.000	17.	3.	2.2	23.50	0.0	0.000
14.	4.	1.9	23.25	0.1	0.004	17.	4.	2.2	23.50	0.0	0.000
14.	5.	1.9	23.25	0.7	0.030	17.	5.	2.2	23.50	0.9	0.038
14.	6.	1.9	23.25	0.1	0.004	17.	6.	2.2	23.50	0.1	0.004
14.	7.	1.9	23.25	0.3	0.013	17.	7.	2.2	23.50	0.5	0.021
14.	8.	1.9	23.25	0.2	0.009	17.	8.	2.2	23.50	0.8	0.034
14.	9.	1.9	23.25	0.2	0.009	17.	9.	2.2	23.50	0.3	0.013
14.	10	1.9	23.25	0.2	0.009	17.	10	2.2	23.50	0.3	0.013
14.	11	1.9	23.25	0.1	0.004	17.	11	2.2	23.50	0.0	0.000
14.	12	1.9	23.25	0.1	0.004	17.	12	2.2	23.50	0.1	0.004
14.	13	1.9	23.25	0.1	0.004	17.	13	2.2	23.50	0.0	0.000
14.	14	1.9	23.25	0.5	0.022	17.	14	2.2	23.50	0.1	0.004
14.	15	1.9	23.25	0.1	0.004	17.	15	2.2	23.50	0.0	0.000
14.	16	1.9	23.25	0.1	0.004	17.	16	2.2	23.50	0.0	0.000
14.	17	1.9	23.25	0.1	0.004	17.	17	2.2	23.50	0.0	0.000
15.	1.	1.9	24.50	1.5	0.061	18.	1.	13.5	27.00	13.0	0.481
15.	2.	1.9	24.50	0.8	0.033	18.	2.	13.5	27.00	12.9	0.478
15.	3.	1.9	24.50	0.1	0.004	18.	3.	13.5	27.00	0.9	0.033
15.	4.	1.9	24.50	0.1	0.004	18.	4.	13.5	27.00	0.8	0.030
15.	5.	1.9	24.50	0.4	0.016	18.	5.	13.5	27.00	11.9	0.441
15.	6.	1.9	24.50	0.3	0.012	18.	6.	13.5	27.00	5.8	0.215
15.	7.	1.9	24.50	0.4	0.016	18.	7.	13.5	27.00	12.0	0.444
15.	8.	1.9	24.50	0.4	0.016	18.	8.	13.5	27.00	4.0	0.148
15.	9.	1.9	24.50	0.3	0.012	18.	9.	13.5	27.00	6.0	0.222
15.	10	1.9	24.50	0.2	0.008	18.	10	13.5	27.00	7.9	0.293
15.	11	1.9	24.50	0.1	0.004	18.	11	13.5	27.00	1.1	0.041
15.	12	1.9	24.50	0.1	0.004	18.	12	13.5	27.00	5.0	0.185
15.	13	1.9	24.50	0.2	0.008	18.	13	13.5	27.00	0.1	0.004
15.	14	1.9	24.50	0.5	0.020	18.	14	13.5	27.00	5.0	0.185
15.	15	1.9	24.50	0.0	0.000	18.	15	13.5	27.00	0.1	0.004
15.	16	1.9	24.50	0.0	0.000	18.	16	13.5	27.00	1.0	0.037
15.	17	1.9	24.50	0.0	0.000	18.	17	13.5	27.00	0.1	0.004

APPENDIX A (continued).

WEEK NO.	COL NO.	DIST-WATER APPLIED (LITERS)	RETENTION TIME (HRS)	LEACHATE COLLECTED (LITERS)	LEACHATE FLOW RATE (LITERS/HR)	WEEK NO.	COL NO.	DIST-WATER APPLIED (LITERS)	RETENTION TIME (HRS)	LEACHATE COLLECTED (LITERS)	LEACHATE FLOW RATE (LITERS/HR)
19.	1.	2.2	19.00	2.0	0.105	22.	1.	1.5	24.50	1.0	0.041
19.	2.	2.2	19.00	1.9	0.100	22.	2.	1.5	24.50	0.8	0.033
19.	3.	2.2	19.00	0.1	0.005	22.	3.	1.5	24.50	0.1	0.004
19.	4.	2.2	19.00	0.1	0.005	22.	4.	1.5	24.50	0.1	0.004
19.	5.	2.2	19.00	2.1	0.111	22.	5.	1.5	24.50	1.0	0.041
19.	6.	2.2	19.00	4.5	0.237	22.	6.	1.5	24.50	1.5	0.061
19.	7.	2.2	19.00	1.3	0.068	22.	7.	1.5	24.50	0.9	0.037
19.	8.	2.2	19.00	1.0	0.053	22.	8.	1.5	24.50	0.9	0.037
19.	9.	2.2	19.00	2.7	0.142	22.	9.	1.5	24.50	0.9	0.037
19.	10	2.2	19.00	2.0	0.105	22.	10	1.5	24.50	0.5	0.020
19.	11	2.2	19.00	1.1	0.058	22.	11	1.5	24.50	0.8	0.033
19.	12	2.2	19.00	2.0	0.105	22.	12	1.5	24.50	0.7	0.029
19.	13	2.2	19.00	0.5	0.026	22.	13	1.5	24.50	0.1	0.004
19.	14	2.2	19.00	2.7	0.142	22.	14	1.5	24.50	1.2	0.049
19.	15	2.2	19.00	0.0	0.000	22.	15	1.5	24.50	0.1	0.004
19.	16	2.2	19.00	1.0	0.053	22.	16	1.5	24.50	0.1	0.004
19.	17	2.2	19.00	0.0	0.000	22.	17	1.5	24.50	0.1	0.004
20.	1.	11.3	23.50	10.2	0.434	23.	1.	1.5	23.75	1.0	0.042
20.	2.	11.3	23.50	9.8	0.417	23.	2.	1.5	23.75	0.9	0.038
20.	3.	11.3	23.50	5.0	0.213	23.	3.	1.5	23.75	0.1	0.004
20.	4.	11.3	23.50	4.7	0.200	23.	4.	1.5	23.75	0.1	0.004
20.	5.	11.3	23.50	10.0	0.426	23.	5.	1.5	23.75	1.0	0.042
20.	6.	11.3	23.50	5.3	0.226	23.	6.	1.5	23.75	1.1	0.046
20.	7.	11.3	23.50	10.4	0.443	23.	7.	1.5	23.75	0.9	0.038
20.	8.	11.3	23.50	7.1	0.302	23.	8.	1.5	23.75	1.0	0.042
20.	9.	11.3	23.50	7.9	0.336	23.	9.	1.5	23.75	0.8	0.034
20.	10	11.3	23.50	6.2	0.264	23.	10	1.5	23.75	0.5	0.021
20.	11	11.3	23.50	5.8	0.247	23.	11	1.5	23.75	0.4	0.017
20.	12	11.3	23.50	6.1	0.260	23.	12	1.5	23.75	0.1	0.004
20.	13	11.3	23.50	4.9	0.209	23.	13	1.5	23.75	0.1	0.004
20.	14	11.3	23.50	6.2	0.264	23.	14	1.5	23.75	1.0	0.042
20.	15	11.3	23.50	4.7	0.200	23.	15	1.5	23.75	0.1	0.004
20.	16	11.3	23.50	7.2	0.306	23.	16	1.5	23.75	0.1	0.004
20.	17	11.3	23.50	3.9	0.166	23.	17	1.5	23.75	0.1	0.004
21.	1.	1.5	25.75	2.0	0.078	24.	1.	12.8	22.50	12.5	0.556
21.	2.	1.5	25.75	1.1	0.043	24.	2.	12.8	22.50	11.5	0.511
21.	3.	1.5	25.75	0.0	0.000	24.	3.	12.8	22.50	3.7	0.164
21.	4.	1.5	25.75	0.1	0.004	24.	4.	12.8	22.50	4.0	0.178
21.	5.	1.5	25.75	2.0	0.078	24.	5.	12.8	22.50	11.6	0.516
21.	6.	1.5	25.75	5.0	0.194	24.	6.	12.8	22.50	3.2	0.142
21.	7.	1.5	25.75	1.2	0.047	24.	7.	12.8	22.50	11.9	0.529
21.	8.	1.5	25.75	1.1	0.043	24.	8.	12.8	22.50	1.0	0.044
21.	9.	1.5	25.75	2.0	0.078	24.	9.	12.8	22.50	7.1	0.316
21.	10	1.5	25.75	1.9	0.074	24.	10	12.8	22.50	6.5	0.289
21.	11	1.5	25.75	2.1	0.082	24.	11	12.8	22.50	4.0	0.178
21.	12	1.5	25.75	2.5	0.097	24.	12	12.8	22.50	6.5	0.289
21.	13	1.5	25.75	2.0	0.078	24.	13	12.8	22.50	2.0	0.089
21.	14	1.5	25.75	1.7	0.066	24.	14	12.8	22.50	2.0	0.089
21.	15	1.5	25.75	0.1	0.004	24.	15	12.8	22.50	4.2	0.187
21.	16	1.5	25.75	0.5	0.019	24.	16	12.8	22.50	8.0	0.356
21.	17	1.5	25.75	0.7	0.027	24.	17	12.8	22.50	4.5	0.200

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APPENDIX B

COLUMN LEACHATE PHYSICAL AND INORGANIC DATA

EXPLANATION OF DATA:

COL NO. = Column number ranging from 1 to 17.

WEEK NO. = Week of study period ranging from 1 to 24.

TEMP DEG-C = Leachate temperature, °C.

TURB NTU = Leachate turbidity measured in nephelometric turbidity units (NTU).

COND UMO/CM = Leachate specific conductance, umhos/cm at 25°C.

ACID MG/L = Acidity, mg/l.

NOTE: Blank spaces in the data set indicate instances when data were not obtained due to insufficient sample volumes (see Appendix A) or instrumentation problems occured.

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APPENDIX B (continued).

COL NO.	WEEK NO.	TEMP DEG-C	PH	TURB NTU	COND UMO/CM	ACID MG/L	COL NO.	WEEK NO.	TEMP DEG-C	PH	TURB NTU	COND UMO/CM	ACID MG/L
1.	1.	34.	1.3	22.	36600.	60500.	1.	4.	31.	1.7	27.	42800.	49500.
2.	1.	34.	1.3	40.	37200.	69400.	2.	4.	30.	1.7	26.	39500.	52200.
3.	1.	33.	6.4	200.	1080.	11600.	3.	4.	31.	6.5	200.	1460.	320.
4.	1.	34.	6.3	200.	973.	10900.	4.	4.	30.	6.5	200.	1460.	320.
5.	1.	33.	1.4	24.	32800.	45600.	5.	4.	30.	1.8	35.	33500.	35500.
6.	1.	33.	1.4	22.	35400.	77800.	6.	4.	30.	1.7	33.	38000.	58300.
7.	1.	33.	1.4	140.	31700.	56800.	7.	4.	30.	1.7	38.	36700.	48600.
8.	1.	34.	1.4	85.	31200.	52600.	8.	4.	30.	1.7	37.	34300.	41000.
9.	1.	32.	1.2	100.	35900.	110000.	9.	4.	30.	1.6	13.	41100.	81800.
10.	1.	33.	1.4	8.9	34400.	84200.	10.	4.	30.	1.7	45.	34300.	40700.
11.	1.	33.	1.4	40.	35000.	62900.	11.	4.	30.	1.6	45.	36700.	53400.
12.	1.	33.	1.4	40.	30400.	48300.	12.	4.	30.	1.9	34.	27600.	28800.
13.	1.	32.	1.4	27.	31200.	50100.	13.	4.	29.	1.9	30.	24400.	23900.
14.	1.	33.	1.4	13.	32500.	58600.	14.	4.	29.	1.7	28.	35400.	52800.
15.	1.	32.	1.5	30.	26400.	40200.	15.	4.	30.	1.8	13.	35800.	64800.
16.	1.	33.	1.5	16.	27600.	49700.	16.	4.	30.	1.8	19.	29100.	44100.
17.	1.	32.	1.4	65.	28800.	66300.	17.	4.	29.	1.7	12.	36200.	68700.
1.	2.	28.	1.5	30.	43500.	52200.	1.	5.	30.	1.5	26.	35800.	48200.
2.	2.	28.	1.5	26.	43500.	59000.	2.	5.	29.	1.5	23.	34500.	50500.
3.	2.	28.	6.0	200.	1840.	11400.	3.	5.	29.	6.3	200.	1380.	916.
4.	2.	27.	6.2	200.	1920.	11800.	4.	5.	29.	6.4	200.	1460.	720.
5.	2.	27.	1.6	30.	37800.	40900.	5.	5.	29.	1.9	45.	24100.	33500.
6.	2.	27.	1.5	30.	40500.	62300.	6.	5.	29.	1.5	35.	36400.	52400.
7.	2.	27.	1.6	45.	37600.	49000.	7.	5.	28.	1.6	36.	30500.	45700.
8.	2.	26.	1.6	45.	35400.	41400.	8.	5.	29.	1.6	38.	31500.	40000.
9.	2.	27.	1.5	45.	42000.	91200.	9.	5.	28.	1.6	15.	36400.	82400.
10.	2.	27.	1.6	40.	38900.	54600.	10.	5.	29.	1.7	50.	29600.	36800.
11.	2.	27.	1.6	35.	40800.	54900.	11.	5.	28.	1.6	29.	25300.	50500.
12.	2.	25.	1.6	35.	33400.	34800.	12.	5.	29.	2.0	50.	21000.	27400.
13.	2.	26.	1.7	45.	30700.	32600.	13.	5.	28.	2.0	50.	19400.	20900.
14.	2.	26.	1.6	60.	39100.	56400.	14.	5.	28.	1.6	26.	31900.	50100.
15.	2.	27.	1.6	30.	35400.	52900.	15.	5.	29.	1.7	13.	31700.	56800.
16.	2.	27.	1.7	25.	31800.	47400.	16.	5.	29.	1.7	25.	25100.	44600.
17.	2.	27.	1.6	8.3	34600.	71300.	17.	5.	28.	1.7	13.	30400.	53900.
1.	3.	35.	1.4	30.	44600.	52500	1.	6.	28.	1.8	19.	37800.	52400.
2.	3.	33.	1.5	25.	39700.	57500	2.	6.	28.	1.9	20.	37800.	48200.
3.	3.	33.	6.3	200.	1780.	374	3.	6.	28.	6.1	200.	1590.	405.
4.	3.	30.	6.2	200.	1760.	986	4.	6.	28.	6.4	200.	1560.	223.
5.	3.	30.	1.6	33.	36900.	37000	5.	6.	27.	2.2	60.	25700.	34800.
6.	3.	30.	1.5	33.	39700.	62300	6.	6.	27.	1.9	32.	38400.	57800.
7.	3.	30.	1.5	41.	35700.	50200	7.	6.	27.	2.0	38.	33900.	47900.
8.	3.	32.	1.6	40.	36900.	41600	8.	6.	27.	1.6	33.	32300.	40500.
9.	3.	30.	1.4	13.	41200.	88300	9.	6.	28.	1.8	15.	37400.	78100.
10.	3.	30.	1.6	45.	38300.	47800	10.	6.	28.	1.8	50.	28100.	31400.
11.	3.	30.	1.6	35.	38300.	52700	11.	6.	27.	1.5	28.	37500.	51700.
12.	3.	30.	1.7	36.	31500.	32900	12.	6.	27.	2.1	50.	22300.	26900.
13.	3.	29.	1.9	45.	26100.	27600	13.	6.	27.	2.2	65.	18200.	20800.
14.	3.	30.	1.6	40.	36900.	54300	14.	6.	27.	1.5	25.	34900.	50400.
15.	3.	30.	1.6	12.	36900.	58400	15.	6.	29.	1.6	17.	33800.	59400.
16.	3.	30.	1.8	20.	34600.	47100	16.	6.	28.	1.7	22.	29800.	43900.
17.	3.	30.	1.6	8.8	35700.	72500	17.	6.	27.	1.6	12.	29900	47900.

APPENDIX B (continued).

COL NO.	WEEK NO.	TEMP DEG-C	PH	TURB NTU	COND UMO/CM	ACID MG/L	COL NO.	WEEK NO.	TEMP DEG-C	PH	TURB NTU	COND UMO/CM	ACID MG/L
1.	7.	24.	1.2	18.	36900.	49100.	1.	10.	26.	1.4	8.3	44000.	82700.
2.	7.	24.	1.6	17.	35000.	46100.	2.	10.	28.	1.3	9.5	43600.	84300.
3.	7.	24.	7.3	200.	1480.	1380.	3.	10.	30.	6.8	200.	1600.	1080.
4.	7.	24.	7.1	200.		914.	4.	10.	31.	6.8	200.	1640.	880.
5.	7.	24.	2.8	90.	21700.	34200.	5.	10.	29.	2.3	8.8	24700.	24800.
6.	7.	24.	1.6	27.	36200.	52400.	6.	10.	29.	2.0	9.6	30000.	32200.
7.	7.	24.	1.3	29.	31900.	45100.	7.	10.	29.	1.8	6.3	27700.	22600.
8.	7.	23.	1.4	27.	31600.	40600.	8.	10.	29.	1.7	9.7	32300.	31200.
9.	7.	24.	1.6	17.	36000.	76900.	9.	10.	30.	1.8	19.	31500.	35100.
10.	7.	24.	1.7	60.	24000.	31200.	10.	10.	30.	2.7	33.	17400.	21100.
11.	7.	24.	1.3	26.	35300.	52400.	11.	10.	30.	1.8	30.	34200.	39200.
12.	7.	24.	2.0	80.	21000.	27500.	12.	10.	31.	2.7	25.	17500.	18100.
13.	7.	24.	2.1	75.	15700.	18200.	13.	10.	29.	2.9	20.	10800.	11300.
14.	7.	24.	1.3	29.	31200.	46800.	14.	10.	31.	1.9	20.	24900.	26600.
15.	7.	24.	1.4	23.		55800.	15.	10.	28.	2.0	23.	21400.	21700.
16.	7.	24.	1.5	32.		42100.	16.	10.	27.	2.2	24.	14800.	13500.
17.	7.	24.	1.3	14.	26000.	37400.	17.	10.	26.	2.3	10.	11400.	8780.
1.	8.	24.	1.6	14.	39500.	47500.	1.	11.	26.	1.3	5.9	43500.	90900.
2.	8.	24.	1.7	15.	34700.	40100.	2.	11.	26.	1.4	9.5	43000.	95200.
3.	8.						3.	11.	26.	7.8	200.	1480.	11500.
4.	8.	23.	7.8	200.		782.	4.	11.	26.	7.9	200.	1480.	11500.
5.	8.	24.	2.5	35.	26600.	33900.	5.	11.	26.	2.4	11.	16600.	15100.
6.	8.	24.	1.7	23.	35400.	49000.	6.	11.	27.	2.1	18.	27400.	33700.
7.	8.	23.	1.7	21.	34500.	39300.	7.	11.	27.	1.7	13.	28000.	27100.
8.	8.	22.	1.8	23.	34000.	41700.	8.	11.	26.	1.8	8.0	28400.	25400.
9.	8.	24.	1.7	24.		67500.	9.	11.	27.	1.9	12.	27600.	41700.
10.	8.	24.	2.4	100.		28100.	10.	11.	27.	2.7	31.	16700.	19100.
11.	8.	24.	1.8	24.	35800.	45900.	11.	11.	27.	2.0	19.	31400.	44900.
12.	8.	22.	2.5	80.		24200.	12.	11.	27.	2.7	40.	16000.	19300.
13.	8.	23.	2.6	77.		13600.	13.	11.	27.	2.8	22.	9590.	19000.
14.	8.	23.	1.7	27.	32300.	42600.	14.	11.	27.	2.2	13.	20300.	20300.
15.	8.	23.	1.8	26.		48700.	15.	11.					
16.	8.	23.	1.9			36100.	16.	11.	25.	2.5	75.	11300.	14400.
17.	8.	23.	1.8	15.	24900.	29100.	17.	11.	25.	2.3	8.7	10550.	7610.
1.	9.	23.	1.5	6.8	37400.	49000.	1.	12.	20.	1.3	4.6	44300.	79200.
2.	9.	23.	1.5	5.9	36500.	49600.	2.	12.	21.	1.4	5.1	43100.	85600.
3.	9.	23.	6.7	200.	1850.	261.	3.	12.	23.				783.
4.	9.	22.	6.7	200.	1900.	392.	4.	12.					1080.
5.	9.	22.	2.4	19.	27100.	32800.	5.	12.	21.	2.5	4.6	10700.	7610.
6.	9.	22.	1.7	15.	35200.	45000.	6.	12.	22.	2.4	14.	22400.	28700.
7.	9.	22.	1.8	7.2	29400.	29900.	7.	12.	21.	1.7	6.7	25900.	23100.
8.	9.	21.	1.8	20.	32000.	36200.	8.	12.	20.	1.8	6.5	25000.	16900.
9.	9.	23.	1.6	12.	36400.	53900.	9.	12.	23.	2.0	15.	27800.	39000.
10.	9.	23.	2.5	60.	21500.	23000.	10.	12.	23.	2.7	50.	15700.	20400.
11.	9.	24.	1.8	27.	34400.	44200.	11.	12.	23.	2.4	75.		46300.
12.	9.	22.	2.6	60.	21000.	23300.	12.	12.	24.	2.6	90.	14700.	18400.
13.	9.	23.	2.7	60.	14300.	13400.	13.	12.	23.	2.8	24.	10600.	9070.
14.	9.	23.	1.7	60.	32100.	35700.	14.	12.	23.	2.2	12.	19900.	20100.
15.	9.	22.	2.0	11.	22400.	21700.	15.	12.	21.	2.2	34.		26300.
16.	9.	22.	2.1	10.	16700.	15100.	16.	12.					15700.
17.	9.	22.	2.2	5.3		11000.	17.	12.					10600.

APPENDIX B (continued).

COL NO.	WEEK NO.	TEMP DEG-C	PH	TURB NTU	COND UMO/CM	ACID MG/L	COL. NO.	WEEK NO.	TEMP DEG-C	PH	TURB NTU	COND UMO/CM	ACID MG/L
1.	13.	18.	1.6	4.9	40200.	75000.	1.	16.	25.	1.6	5.2	35100.	54900.
2.	13.	19.	1.5	11.	40000.	76100.	2.	16.	27.	1.6	7.0	34900.	57500.
3.	13.						3.	16.					
4.	13.						4.	16.					
5.	13.	19.	2.7	5.4	8030.	4260.	5.	16.	28.	2.8	8.0	6260.	3210.
6.	13.	20.	2.5	21.		26200.	6.	16.	28.	2.6	35.		21000.
7.	13.	19.	1.9	13.	24200.	22000.	7.	16.	28.	2.0	11.	20100.	17600.
8.	13.	17.	2.0	6.7	21200.	16400.	8.	16.	27.	2.4	8.0	16200.	11400.
9.	13.	19.	2.3	16.	22600.	38900.	9.	16.	27.	2.4	16.	21400.	29200.
10.	13.	19.	2.8	40.	14600.	18600.	10.	16.	26.	2.7	38.		15800.
11.	13.	19.	2.4	24.	27500.	47400.	11.	16.	23.				
12.	13.	17.	2.7	30.	14400.	19000.	12.	16.					
13.	13.	19.	2.8	50.	8940.	10600.	13.	16.					
14.	13.	18.	2.5	12.	14700.	18700.	14.	16.	24.	2.6	19.		13300.
15.	13.					25200.	15.	16.					
16.	13.					11600.	16.	16.					
17.	13.						17.	16.					
1.	14.	27.	1.6	6.1	39600.	64400.	1.	17.	22.	1.6	5.7	33200.	47400.
2.	14.	29.	1.6	6.5	38500.	66400.	2.	17.	23.	2.1	9.3	33400.	52100.
3.	14.						3.	17.					
4.	14.	27.					4.	17.					
5.	14.	28.	2.7	8.5	7330.	3900.	5.	17.	24.	2.9	12.	5860.	3240.
6.	14.	28.	2.6	20.	17200.	23300.	6.	17.	24.	2.7	29.	18000.	21500.
7.	14.	29.	1.9	12.	23400.	20600.	7.	17.	24.	2.1	20.	18600.	15600.
8.	14.	28.	2.0	9.5	20300.	12600.	8.	17.	24.	2.5	7.7	15100.	10800.
9.	14.	28.	2.3	18.5	22200.	33000.	9.	17.	24.	2.5	25.	19700.	29100.
10.	14.	27.	2.7	45.	14000.	16900.	10.	17.	24.	2.8	33.	14000.	16100.
11.	14.	27.	2.6	50.		39900.	11.	17.					
12.	14.	27.	2.7	34.		24500.	12.	17.	24.	2.8	25.	14800.	16500.
13.	14.	27.	2.7	23.		9320.	13.	17.					
14.	14.	27.	2.5	15.	14200.	15000.	14.	17.	23.	2.7	25.	12900.	14400.
15.	14.	21.	2.4			18200.	15.	17.					
16.	14.	20.	2.4				16.	17.					
17.	14.	20.	2.5			8020	17.	17.					
1.	15.	21.	1.5	6.0	34400.	64500.	1.	18.	25.	1.6	5.1	26800.	43200.
2.	15.	22.	1.6	6.5	32800.	64700.	2.	18.	26.	1.6	6.6	24400.	44300.
3.	15.						3.	18.	30.	7.0	14.	683.	972.
4.	15.						4.	18.	31.	7.5	30.	699.	1300.
5.	15.	23.	2.8	7.9	6000.	3690.	5.	18.	28.	3.1	1.6	5910.	4730.
6.	15.	23.	2.5	65.		23900.	6.	18.	29.	3.0	5.5	11200.	11600.
7.	15.	23.	1.9	12.	19600.	21700.	7.	18.	27.	2.2	2.7	13500.	16100.
8.	15.	23.	2.1	8.8	14800.	12400.	8.	18.	28.	2.5	2.8	10800.	10500.
9.	15.	23.	2.3	18.	11600.	34300.	9.	18.	28.	2.3	5.2	14300.	19900.
10.	15.	23.	2.7	34.	11400.	17100.	10.	18.	27.	2.7	13.	10600.	12300.
11.	15.	23.	2.5	100.		48100.	11.	18.	28.	2.7	14.	20700.	34000.
12.	15.	24.	2.7	26.	11100.	17100.	12.	18.	26.	3.2	9.9	9970.	12700.
13.	15.	23.	2.7	36.	8030.	9660.	13.	18.					
14.	15.	23.	2.6	16.	11400.	15000.	14.	18.	27.		6.4	9410.	
15.	15.						15.	18.					
16.	15.						16.	18.	22.	2.0	24.	12100.	22000.
17.	15.						17.	18.					

(PREVIOUSLY CIN. EPA FORM 287)

APPENDIX B (continued).

COL NO.	WEEK NO.	TEMP DEG-C	PH	TURB NTU	COND UMO/CM	ACID MG/L	COL NO.	WEEK NO.	TEMP DEG-C	PH	TURB NTU	COND UMO/CM	ACID MG/L
1.	19.	22.	1.4	.90	36200.	78400.	1.	22.	12.	1.4	.80	42100.	81600.
2.	19.	22.	1.4	1.1	35800.	74200.	2.	22.	13.	1.5	.70	48100.	80600.
3.	19.		4.4				3.	22.					
4.	19.		7.4				4.	22.					
5.	19.	22.	3.1	16.	7440.	5570.	5.	22.	14.	3.0	6.5	6080.	3360.
6.	19.	22.	3.0	3.6	7500.	5210.	6.	22.	14.	3.1	3.1	7130.	4080.
7.	19.	22.	1.8	2.2	22200.	32000.	7.	22.	14.	1.8	2.6	25200.	26200.
8.	19.	21.	2.0	2.6	19000.	22200.	8.	22.	14.	1.7	1.3	24100.	26400.
9.	19.	22.	2.1	6.8	14600.	17200.	9.	22.	14.	1.9	2.3	24100.	42000.
10.	19.	22.	1.9	2.7	19200.	28800.	10.	22.	14.	1.6	2.1	29100.	47500.
11.	19.	22.	2.4	7.5	19600.	28100.	11.	22.	14.	1.6	.90	30000.	47300.
12.	19.	22.	2.9	6.5	9510.	10600.	12.	22.	14.	1.8	.90	25300.	47500.
13.	19.	22.	2.9	19.	8980.	9360.	13.	22.	13.	2.8	15.	7940.	7080.
14.	19.	21.	2.9	10.	8260.	8040.	14.	22.	13.	3.1	5.4	7890.	6480.
15.	19.						15.	22.		2.2			
16.	19.	20.	2.0	13.	10500.	12700.	16.	22.		2.9			
17.	19.						17.	22.					
1.	20.	24.	1.7	2.8	33900.	50000.	1.	23.	14.	1.4	.75	42500.	69800.
2.	20.	26.	1.7	3.5	33400.	48700.	2.	23.	17.	1.4	.60	43200.	80900.
3.	20.	28.	6.0	3.7	1150.	960.	3.	23.					
4.	20.	28.	6.4	4.7	1040.	960.	4.	23.					
5.	20.	27.	3.2	7.6	5340.	3240.	5.	23.	17.	3.1	4.8	5720.	3120.
6.	20.	28.	3.2	1.6	7360.	4800.	6.	23.	17.	3.1	2.9	7120.	3600.
7.	20.	26.	2.0	1.9	22500.	23500.	7.	23.	17.	1.8	2.8	23500.	22200.
8.	20.	26.	2.1	1.7	18800.	18100.	8.	23.	18.	1.8	1.5	24400.	24000.
9.	20.	27.	2.2	2.1	15600.	17000.	9.	23.	17.	1.7	1.2	36700.	51400.
10.	20.	28.	2.2	5.3	19100.	23600.	10.	23.	17.	1.6	1.9	26700.	52800.
11.	20.	28.	2.2	4.2	21400.	26900.	11.	23.	17.	1.5	1.3	26700.	52200.
12.	20.	28.	3.1	4.2	12100.	13300.	12.	23.	18.	1.6	2.2	24300.	50900.
13.	20.	28.	3.1	4.1	8460.	8160.	13.	23.					
14.	20.	27.	3.0	1.5	8040.	6720.	14.	23.	16.	2.8	4.9		6720.
15.	20.	21.	2.2	6.0	13000.	12000.	15.	23.					
16.	20.	21.	2.5	1.6	6250.	4320.	16.	23.					
17.	20.	21.	2.3	5.7	8100.	5400.	17.	23.					
1.	21.	15.	1.4	.70	53400.	81500.	1.	24.	13.	1.6	1.3	27900.	31000
2.	21.	16.	1.4	.80	50800.	75400.	2.	24.	14.	1.5	1.7	31500.	35800
3.	21.						3.	24.	14.	5.0	7.0	2270.	284
4.	21.	16.	7.0	13.	1270.	960.	4.	24.	14.	6.1	3.8	1680.	50
5.	21.	17.	2.9	4.3	9130.	3000.	5.	24.	14.	3.3	11.	4130.	2430
6.	21.	16.	3.1	1.9	7860.	4080.	6.	24.	15.	3.3	3.2	5530.	3090
7.	21.	17.	1.8	2.2	27100.	26600.	7.	24.	15.	1.9	1.7	17600.	13600
8.	21.	15.	1.8	1.2	23400.	31100.	8.	24.	13.	1.9	1.7	15900.	11200
9.	21.	17.	1.8	2.0	26700.	32200.	9.	24.	14.	1.8	.85	21600.	31800
10.	21.	17.	1.7	1.2	32400.	48500.	10.	24.	15.	1.7	1.4	23900.	34700
11.	21.	17.	1.7	.90	34000.	42500.	11.	24.	15.	1.6	.70	26800.	37100
12.	21.	16.	1.8	.50	27000.	42700.	12.	24.	14.	1.8	1.4	21200.	27700
13.	21.	16.	2.8	4.7	8410.	5760.	13.	24.	14.	2.8	4.1	7160.	5920
14.	21.	16.	2.9	3.6	8440.	6000.	14.	24.	14.	3.0	20.	6830.	5530
15.	21.	15.	2.2	9.9	14800.	13700.	15.	24.	13.	2.0	3.3	10100.	8290
16.	21.	15.	2.2	2.8	10000.	6720.	16.	24.	13.	2.4	12.	4480.	2980
17.	21.	14.	2.1	3.1	11400.	6840.	17.	24.	12.	2.3	2.0	5680.	3180

APPENDIX C

COLUMN LEACHATE METAL CONCENTRATION DATA

EXPLANATION OF DATA SYMBOLS:

COL NO. = Column number, ranging from 1 to 17.

WEEK NO. = Week of study period, ranging from 1 to 24.

CU MG/L = Leachate total copper concentration, mg/l.

FE MG/L = Leachate total iron concentration, mg/l.

PB MG/L = Leachate total lead concentration, mg/l.

ZN MG/L = Leachate total zinc concentration, mg/l.

NOTE: Blank spaces in the data set indicate instances when data were not obtained due to insufficient sample volumes (see Appendix A) or instrumentation problems occured.

APPENDIX C (continued).

COL NO.	WEEK NO.	CU MG/L	FE MG/L	PB MG/L	ZN MG/L
1.	1.	299.	23400.	3.69	210.
2.	1.	352.	27200.	4.34	239.
3.	1.	.766	427.	.48	.884
4.	1.	.885	336.	.26	1.17
5.	1.	421.	15900.	3.09	216.
6.	1.	288.	31200.	2.71	279.
7.	1.	287.	22300.	3.12	257.
8.	1.	310.	20200.	2.32	234.
9.	1.	321.	37900.	2.75	240.
10.	1.	229.	31100.	2.47	223.
11.	1.	266.	23700.	2.99	234.
12.	1.	384.	18500.	2.21	266.
13.	1.	254.	20400.	2.07	220.
14.	1.		21600.	1.87	299.
15.	1.		13600.	1.10	182.
16.	1.		18600.	1.59	193.
17.	1.		24800.	1.69	193.
1.	2.	289.	20300.	3.41	191.
2.	2.	299.	22700.	3.79	202.
3.	2.	20.7	1500.	.267	25.9
4.	2.	1.32	290.	.135	10.
5.	2.	402.	13800.	3.07	209.
6.	2.	253.	23400.	2.45	235.
7.	2.	256.	19800.	1.74	239.
8.	2.	275.	17500.	2.04	208.
9.	2.	296.	17100.	2.37	225.
10.	2.	172.	21400.	2.43	151.
11.	2.	240.	20700.	1.79	208.
12.	2.	299.	13000.	1.35	200.
13.	2.	179.	13900.	1.48	154.
14.	2.	334.	19700.	3.18	296.
15.	2.	286.	19800.	.896	246.
16.	2.	210.	17600.	.769	179.
17.	2.	239.	14100.	.675	200.
1.	3.	557.	38400.	3.14	358.
2.	3.	613.	43800.		394.
3.	3.	.238	1310.		2.45
4.	3.	.344	1080.		5.11
5.	3.	854.	29300.	2.75	381.
6.	3.	490.	45400.	1.69	446.
7.	3.	473.	50900.	.376	376.
8.	3.	543.	32500.	1.51	373.
9.	3.	576.	65900.	.866	346.
10.	3.	287.	33200.	2.05	275.
11.	3.	459.	36000.	2.41	370.
12.	3.	512.	22100.	1.25	336.
13.	3.	255.	20000.	1.36	232.
14.	3.	712.	40700.	2.02	579.
15.	3.	607.	39500.	.756	491.
16.	3.	424.	33500.	.666	309.
17.	3.	592.	63900.	.739	444.

COL NO.	WEEK NO.	CU MG/L	FE MG/L	PB MG/L	ZN MG/L
1.	4.	476.	32000.	2.64	313.
2.	4.	507.	35500.	2.56	340.
3.	4.	.944	1180.		10.1
4.	4.	.113	1070.		7.91
5.	4.	741.	26700.	1.70	345.
6.	4.	449.	40400.	.780	429.
7.	4.	465.	34900.	1.47	281.
8.	4.	447.	27200.	1.69	327.
9.	4.	560.	61300.	1.44	415.
10.	4.	222.	27700.	1.93	220.
11.	4.	421.	30900.	2.36	368.
12.	4.	479.	21200.	.478	385.
13.	4.	208.	17600.	1.30	241.
14.	4.	617.	34100.	1.60	496.
15.	4.	713.	48500.	.829	598.
16.	4.	390.	32200.	.258	336.
17.	4.	401.	42900.	.393	305.
1.	5.	266.	13800.	3.02	160.
2.	5.	270.	17900.	3.10	159.
3.	5.	.201	497.	.474	10.2
4.	5.	319.	11800.	2.09	149.
5.	5.	319.	11800.	2.09	149.
6.	5.	226.	17000.	2.11	189.
7.	5.	228.	14400.	1.62	188.
8.	5.	248.	13900.	1.83	153.
9.	5.	273.	19900.	1.36	186.
10.	5.	107.	11200.	1.75	101.
11.	5.	216.	14500.	2.52	172.
12.	5.	221.	7220.	.722	145.
13.	5.	94.0	5080.	.082	100.
14.	5.	320.	15100.	1.85	234.
15.	5.	324.	17200.	1.04	240.
16.	5.	314.	13600.	1.03	160.
17.	5.	209.	18900.	1.41	149.
1.	6.	316.	21100.	3.04	226.
2.	6.	293.	21500.	3.18	206.
3.	6.	.255	4490.	.098	97.6
4.	6.	.155	4450.	.115	21.9
5.	6.	223.	19200.	1.87	238.
6.	6.	243.	21100.	2.38	227.
7.	6.	243.	18700.	1.87	218.
8.	6.	275.	17200.	2.08	200.
9.	6.	281.	32000.	1.31	246.
10.	6.	91.1	15600.	1.91	126.
11.	6.	246.	20600.	2.51	215.
12.	6.	196.	15200.	1.27	192.
13.	6.	74.3	11600.	1.46	126.
14.	6.	316.	19300.	1.78	342.
15.	6.	365.	21100.	1.02	399.
16.	6.	226.	16100.	.809	206.
17.	6.	168.	16900.	.780	125.

APPENDIX C (continued).

COL NO.	WEEK NO.	CU MG/L	FE MG/L	PB MG/L	ZN MG/L
1.	7.	550.	32000.	3.72	371.
2.	7.	463.	31800.	3.67	323.
3.	7.	.226	1570.	.116	8.06
4.	7.	.332	687.	.149	10.7
5.	7.	132.	12800.	1.37	169.
6.	7.	433.	35400.	2.73	412.
7.	7.	425.	30500.	2.35	673.
8.	7.	316.	15000.	2.35	187.
9.	7.	575.	52000.	1.78	421.
10.	7.	198.	21300.	1.64	173.
11.	7.	283.	17100.	2.93	232.
12.	7.	311.	16500.	0.878	331.
13.	7.	136.	10700.	1.79	138.
14.	7.	647.	29100.	2.10	476.
15.	7.	615.	34400.	1.43	535.
16.	7.	373.	26000.	1.24	334.
17.	7.	242.	23700.	1.15	188.
1.	8.	276.	16000.	2.96	177.
2.	8.	219.	16300.	3.02	158.
3.	8.	.255	1150.		.2
4.	8.	.569	258.		.6
5.	8.	68.2	15800.	2.55	180.
6.	8.	176.	17500.	1.88	209.
7.	8.	198.	14600.	1.69	181.
8.	8.	276.	16000.	1.53	196.
9.	8.	235.	25200.	.95	171.
10.	8.	52.0	13500.	1.38	171.
11.	8.	188.	16300.	1.95	201.
12.	8.	63.5	13400.	.82	133.
13.	8.	18.8	6880.	1.77	49.9
14.	8.	257.	13800.	1.15	191.
15.	8.	299.	17400.	.52	221.
16.	8.	166.	12500.	.30	131.
17.	8.	80.2	9970.	.16	79.5
1.	9.	244.	18600.	3.12	169.
2.	9.	180.	16000.	3.30	153.
3.	9.	1.0	1200.		.9
4.	9.	1.3	319.		1.0
5.	9.	156.	13500.	3.38	165.
6.	9.	163.	18100.	1.78	174.
7.	9.	131.	14600.	1.47	122.
8.	9.	210.	16400.	1.66	154.
9.	9.	167.	19400.	.64	127.
10.	9.	121.	10500.	1.66	604.
11.	9.	197.	15800.	1.89	164.
12.	9.	1.8	9840.	.78	85.9
13.	9.	38.5	5910.	1.25	36.2
14.	9.	222.	11900.	1.39	163.
15.	9.	100.	7150.	.20	82.1
16.	9.	41.8	5930.	.10	44.6
17.	9.	18.0	4170.		11.4

COL NO.	WEEK NO.	CU MG/L	FE MG/L	PB MG/L	ZN MG/L
1.	10.	266.	30400.	3.90	29.6
2.	10.	268.	31300.	3.92	307.
3.	10.	.010	1210.		4.66
4.	10.		347.		6.09
5.	10.	264.	11300.	2.93	113.
6.	10.	103.	16100.	1.61	101.
7.	10.	118.	11400.	1.39	76.1
8.	10.	242.	13400.	1.71	133.
9.	10.	183.	22100.	.89	
10.	10.	7.78	10400.	1.39	107.
11.	10.	169.	17600.	2.24	204.
12.	10.	5.13	10200.	.44	97.0
13.	10.	3.48	5180.	1.35	45.6
14.	10.	148.	10900.	1.08	133.
15.	10.	112.	9090.	.36	108.
16.	10.	40.5	6970.	.13	51.0
17.	10.	12.8	3450.		30.2
1.	11.	234.	29400.	3.71	247.
2.	11.	244.	28900.	3.65	407.
3.	11.	.299	175.		.381
4.	11.		210.		.039
5.	11.	116.	5480.	1.68	66.6
6.	11.	73.7	14100.	1.04	134.
7.	11.	88.0	10300.	1.11	53.5
8.	11.	150.	10800.	1.20	76.0
9.	11.	125.	15400.	.65	113.
10.	11.	2.91	9170.	1.30	49.6
11.	11.	153.	17100.	1.82	210.
12.	11.	3.10	10500.	.30	82.3
13.	11.	2.15	5060.	1.06	22.8
14.	11.	107.	9600.	.75	101.
15.	11.				
16.	11.	50.6	6860.	.07	52.6
17.	11.	12.6	3500.	.05	14.7
1.	12.	210.	27600.	3.78	149.
2.	12.	237.	27900.	3.51	156.
3.	12.	.470	346.		.096
4.	12.				
5.	12.	57.9	2070.	1.61	24.7
6.	12.	59.0	11300.	1.24	98.8
7.	12.	89.0	7440.	1.36	49.6
8.	12.	111.	5240.	1.53	47.7
9.	12.	128.	14300.	.77	101.
10.	12.	4.47	8070.	1.33	48.9
11.	12.	163.	16000.	1.61	135.
12.	12.	3.95	8220.	.54	75.4
13.	12.	1.44	3810.	.87	25.5
14.	12.	90.5	7060.	.86	85.7
15.	12.	.993	9140.	.58	102.
16.	12.		3230.		
17.	12.	.693		.24	17.9

APPENDIX C (continued).

COL NO.	WEEK NO.	CU MG/L	FE MG/L	PB MG/L	ZN MG/L
1.	13.	162.	23800.	3.65	110.
2.	13.	181.	24400.	3.52	115.
3.	13.				
4.	13.				
5.	13.	29.6	1260.	1.64	13.1
6.	13.	49.0	10100.	1.22	80.1
7.	13.	1.63	7100.	1.38	42.4
8.	13.	84.3	4070.	1.43	29.3
9.	13.	124.	13100.	.66	84.0
10.	13.	6.17	8000.	1.06	46.1
11.	13.	144.	15800.	1.76	113.
12.	13.	1.48	8000.	.56	66.5
13.	13.	1.28	3900.	.77	27.6
14.	13.	71.9	6620.	.80	66.5
15.	13.				
16.	13.				
17.	13.				
1.	14.	145.	24200.	3.38	195.
2.	14.	162.	25300.	3.19	218.
3.	14.	.130	1000.	.09	4.38
4.	14.	.146	145.	.18	.478
5.	14.	22.9	1540.	1.42	11.8
6.	14.	43.0	9560.	.978	106.
7.	14.	82.1	7030.	1.10	51.8
8.	14.	81.1	4480.	1.01	33.7
9.	14.	108.	13000.	.63	115.
10.	14.	4.08	7870.	1.16	54.7
11.	14.	151.	16400.	1.36	202.
12.	14.	114.	8460.	.42	83.6
13.	14.	.616	4090.	.56	34.8
14.	14.	58.9	6760.	.65	76.9
15.	14.	19.9	6140.	.51	105.
16.	14.	38.3	4140.	.22	57.8
17.	14.	7.47	2080.	.19	19.4
1.	15.	141.	28400.	3.34	167.
2.	15.	153.	25000.	3.19	188.
3.	15.	.050	48.		.491
4.	15.				
5.	15.	18.1	989.	1.04	8.00
6.	15.	43.9	14900.	0.98	106.
7.	15.	81.8	10300.	0.89	48.0
8.	15.	67.8	5900.	1.07	30.8
9.	15.	109.	12900.	0.57	102.
10.	15.	3.15	10800.	0.88	50.1
11.	15.	2.46	12700.	1.04	147.
12.	15.	.921	11100.	0.46	73.1
13.	15.	.587	5600.	0.54	39.4
14.	15.	46.3	8720.	0.60	72.9
15.	15.				
16.	15.				
17.	15.				
1.	16.	126.	19100.	3.71	152.
2.	16.	134.	18900.	3.44	188.
3.	16.				
4.	16.				
5.	16.	12.1	616.	1.24	10.2
6.	16.	28.2	8200.	.76	95.
7.	16.	68.4	5990.	1.06	30.8
8.	16.	77.7	3580.	1.12	17.
9.	16.	95.3	10500.	.72	92.
10.	16.	2.96	6620.	.83	44.8
11.	16.	109.	10800.	.65	143.
12.	16.	1.21	6980.	.23	79.1
13.	16.	1.39	1560.		13.3
14.	16.	39.6	5280.	.47	52.4
15.	16.				
16.	16.				
17.	16.				
1.	17.	105.	16600.	3.38	135.
2.	17.	112.	17100.	3.09	144.
3.	17.	.941	22.		1.40
4.	17.	.567	67.	6.34	2.08
5.	17.	6.20	1140.	2.10	5.63
6.	17.	18.0	8700.	1.72	86.1
7.	17.	57.2	5210.	1.36	36.9
8.	17.	52.4	3840.	1.24	18.4
9.	17.	90.8	10400.	1.01	94.2
10.	17.	1.51	10700.	1.04	36.5
11.	17.	36.3	3130.	0.09	40.3
12.	17.	.435	7670.	0.69	57.9
13.	17.				
14.	17.	38.3	6780.	0.94	76.2
15.	17.				
16.	17.				
17.	17.				
1.	18.	84.2	12100.	4.08	104.
2.	18.	75.3	14200.	3.92	130.
3.	18.	.408	16.	0.258	.295
4.	18.	.376	17.	.346	.269
5.	18.	1.12	3140.	1.99	5.31
6.	18.	2.49	5520.	1.57	38.2
7.	18.	31.6	5740.	2.25	51.0
8.	18.	18.2	4290.	1.80	16.2
9.	18.	40.6	6780.	1.32	42.5
10.	18.	21.3	4280.	2.92	32.7
11.	18.	43.7	11500.	2.83	150.
12.	18.	.765	5530.	1.08	38.9
13.	18.				
14.	18.	12.0	4260.	1.28	29.3
15.	18.				
16.	18.	44.9	6840.	.931	44.4
17.	18.				

APPENDIX C (continued).

COL NO.	WEEK NO.	CU MG/L	FE MG/L	PB MG/L	ZN MG/L
1.	19.	153.	26400.	4.77	302.
2.	19.	135.	24000.	5.62	266.
3.	19.	.147	9.	.398	6.24
4.	19.	.913	10.	.354	2.14
5.	19.	.892	2080.	2.19	19.6
6.	19.	1.38	3170.	1.31	24.0
7.	19.	68.3	3720.	1.69	207.
8.	19.	80.6	9000.	1.44	63.1
9.	19.	58.8	8510.	1.10	43.5
10.	19.	72.8	10500.	2.24	91.6
11.	19.	97.5	10900.	1.87	166.
12.	19.	.223	6740.	1.39	49.1
13.	19.	.550	5530.	1.43	
14.	19.	8.20	3760.	1.30	28.6
15.	19.				
16.	19.	33.7	3920.	1.16	60.6
17.	19.				
1.	20.	68.9	18900.	3.94	143.
2.	20.	55.6	19800.	4.65	166.
3.	20.	1.72	2570.	.075	25.
4.	20.	.37	1590.	.086	15.1
5.	20.	.646	2090.	1.87	9.5
6.	20.	1.17	2770.	1.26	17.6
7.	20.	59.4	8140.	1.91	120.
8.	20.	69.2	8830.	1.58	69.2
9.	20.	45.7	6360.	.77	40.7
10.	20.	75.1	7950.	1.83	73.1
11.	20.	146.	9990.	1.56	186.
12.	20.	25.6	7430.	1.11	86.8
13.	20.	4.13	4500.	2.39	36.7
14.	20.	4.52	3140.	.83	20.
15.	20.				
16.	20.	8.23	880.	.276	17.
17.	20.	14.9	1100.	.441	20.
1.	21.	129.	28500.	4.2	273.
2.	21.	109.	25700.	4.93	261.
3.	21.	.097			13.2
4.	21.	3.06	2170.	.14	49.4
5.	21.	.756	392.	1.84	2.91
6.	21.	10.8	3300.	1.32	17.7
7.	21.	24.9	4100.	1.92	53.5
8.	21.	73.4	11400.	1.28	211.
9.	21.	101.	13000.	.930	137.
10.	21.	97.1	13100.	2.13	159.
11.	21.	76.2	21300.	1.56	269.
12.	21.	161.	16000.	.916	251.
13.	21.	377.	22500.	2.54	70.5
14.	21.	14.4	4600.	.654	26.3
15.	21.	87.1	6800.	.578	86.4
16.	21.	22.	11700.	.263	26.3
17.	21.	25.3	1940.	.286	88.4
1.	22.	120.	24800.	4.65	209.
2.	22.	103.	22600.	4.51	208.
3.	22.	.133	31.		.791
4.	22.	.143	9.	.151	.827
5.	22.	.127	1200.	1.67	14.9
6.	22.	.661	1030.	1.36	5.62
7.	22.	41.7	8390.	2.19	147.
8.	22.	77.8	9590.	1.76	74.9
9.	22.	106.	12000.	1.38	135.
10.	22.	64.2	14900.	2.61	242.
11.	22.	152.	16300.	2.20	334.
12.	22.	214.	15000.	1.35	286.
13.	22.	18.6	3520.	1.89	9.60
14.	22.	3.44	798.	.887	11.5
15.	22.	76.9	3830.	.848	22.4
16.	22.	23.4	2310.	.431	26.5
17.	22.	10.2	639.	.267	9.14
1.	23.	87.9	19800.	3.84	164.
2.	23.	92.5	21200.	3.92	204.
3.	23.				
4.	23.				
5.	23.		1030.	1.20	9.29
6.	23.	.20	1110.	0.76	7.62
7.	23.	29.9	7200.	1.68	118.
8.	23.	45.8	6340.	1.34	56.5
9.	23.	107.	16600.	1.06	91.5
10.	23.	52.1	17200.	1.88	149.
11.	23.	96.9	16700.	1.60	170.
12.	23.	152.	15100.	1.34	167.
13.	23.				
14.	23.	2.69	4710.	0.68	10.0
15.	23.				
16.	23.				
17.	23.				
1.	24.	49.9	9970.	3.10	103.
2.	24.	43.3	8820.	2.99	92.6
3.	24.				.783
4.	24.				.244
5.	24.		606.	1.42	8.99
6.	24.		586.	0.82	24.1
7.	24.	23.1	3400.	1.92	75.4
8.	24.	23.8	4800.	1.48	54.5
9.	24.	64.3	9920.	1.05	66.5
10.	24.	54.9	11700.	2.11	121.
11.	24.	113.	10600.	1.88	150.
12.	24.	139.	8710.	1.02	115.
13.	24.	13.7	1330.	2.02	27.0
14.	24.	1.53	2590.	0.97	49.1
15.	24.	38.8	1910.	0.23	45.7
16.	24.	10.3	686.	0.19	27.2
17.	24.	9.56	805.	0.73	219.

APPENDIX D

VEGETATIVE METAL CONCENTRATION DATA

EXPLANATION OF DATA:

COL NO. = Column number ranging from 1 to 17

CU MG/G = Total copper concentration, mg/g

FE MG/G = Total iron concentration, mg/g

ZN MG/G = Total zinc concentration, mg/g

APPENDIX D (CONTINUED)

VEGETATIVE METAL CONCENTRATIONS AT THE END OF THE STUDY PERIOD (DRY WEIGHT)

COL NO.	CU MG/G	FE MG/G	PB MG/G	ZN MG/G
1.	*	*	*	*
2.	*	*	*	*
3.	0.087	17.4	0.094	0.654
4.	0.094	14.8	0.103	0.640
5.	1.09	21.8	0.989	0.479
6.	3.72	++	3.19	1.46
7.	0.901	40.1	0.408	0.499
8.	1.32	85.4	4.12	1.11
9.	0.114	11.2	0.071	0.554
10.	0.087	15.5	0.085	0.507
11.	0.115	11.6	0.080	0.358
12.	0.091	16.3	0.130	0.476
13.	0.084	15.0	0.090	0.585
14.	0.095	16.2	0.110	0.562
15.	0.089	12.2	0.087	0.407
16.	0.076	9.78	0.095	0.798
17.	0.109	9.76	0.097	1.409

* INDICATES NO VEGETATIVE GROWTH
++INDICATES NO DATA AVAILABLE

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APPENDIX E

SOIL AND SOLID WASTE METAL MIGRATION DATA

EXPLANATION OF DATA:

MW = Mine waste material

CS = Cover soil material

A-1 = Lime, fertilizer, sludge, mine waste (incorporated materials)

A-2 = Lime, fertilizer, sludge, mine waste (layered materials)

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TYPING GUIDE SHEET

APPENDIX E (continued)

Soil and Solid Waste Metal Migration Data
(Dry Weight, mg/g)

Column Number	Metal	1st 0.31m[1] from top of column	2nd 0.31m from top of column	3rd 0.31m from top of column	4th 0.31m from top of column	5th 0.31m from top of column	6th 0.31m from top of column	7th 0.31m from top of column
	Material	(MW)	(MW)	(MW)	(MW)			
1	Copper	5.4	4.9	3.9	+			
	Iron	150	280	300	+			
	Lead	1.2	1.2	0.60	+			
	Zinc	1.1	1.6	1.1	+			
2	Copper	6.7	5.2	5.0	+			
	Iron	130	340	370	+			
	Lead	0.59	2.1	0.49	+			
	Zinc	1.2	1.2	1.0	+			
	Material	(CS)	(CS)	(CS)	(CS)			
3	Copper	0.17	0.06	0.25	0.10			
	Iron	18	*	15	16			
	Lead	0.11	0.0	0.08	0.08			
	Zinc	0.13	0.13	0.26	0.23			
4	Copper	0.05	0.05	0.12	0.07			
	Iron	14	*	15	19			
	Lead	0.06	0.05	0.08	0.07			
	Zinc	0.14	0.28	0.18	0.16			
	Material	(A-1)	(MW)	(MW)	(MW)			
5	Copper	5.0	4.0	*	3.7			
	Iron	150	280	340	280			
	Lead	1.6	1.8	1.7	1.0			
	Zinc	1.1	2.5	1.2	1.7			

[1] 0.31m is approximately equal to 1 foot.

+ No data available due to material compaction.

* Data not available due to sampling and/or analytical problems.

(continued)

APPENDIX E (continued)

Soil and Solid Waste Metal Migration Data

(Dry Weight, mg/g)

Column Number	Metal	1st 0.31m[1] from top of column	2nd 0.31m from top of column	3rd 0.31m from top of column	4th 0.31m from top of column	5th 0.31m from top of column	6th 0.31m from top of column	7th 0.31m from top of column
6	Copper	2.6	4.5	6.0	4.9			
	Iron	240	230	270	290			
	Lead	*	2.1	2.5	2.2			
	Zinc	1.2	2.5	2.4	1.2			
Material		(A-2)	(MW)	(MW)	(MW)			
7	Copper	3.5	3.2	*	*			
	Iron	220	270	*	*			
	Lead	1.8	2.6	*	*			
	Zinc	1.2	1.9	*	*			
8	Copper	3.2	6.2	4.9	5.1			
	Iron	250	230	260	250			
	Lead	2.0	1.1	2.0	1.6			
	Zinc	1.0	1.2	1.1	1.1			
Material		(CS)	(CS)[2]	(MW)	(MW)	(MW)	(MW)	
9	Copper	*	*	2.3	4.8	1.9	+	
	Iron	150	210	190	*	150	+	
	Lead	0.07	1.8	1.9	1.0	1.2	+	
	Zinc	0.57	0.60	0.46	0.50	1.1	+	
10	Copper	0.07	0.21	3.2	3.2	*	+	
	Iron	14	27	190	230	*	+	
	Lead	0.05	0.16	0.89	0.48	*	+	
	Zinc	0.19	0.14	0.85	0.92	*	+	

[1] 0.31m is approximately equal to 1 foot.

+ No data available due to material compaction.

* Data not available due to sampling and/or analytical problems.

[2] A total 0.47m (1.5 ft) of cover soil was used in these columns. In this case only, this second section is therefore only 0.16m (0.5 ft).

(continued)

APPENDIX E (continued)

Soil and Solid Waste Metal Migration Data

(Dry Weight, mg/g)

Column Number	Metal	1st 0.31m[1] from top of column	2nd 0.31m from top of column	3rd 0.31m from top of column	4th 0.31m from top of column	5th 0.31m from top of column	6th 0.31m from top of column	7th 0.31m from top of column
Material		(CS)	(CS)	(MW)	(MW)	(MW)	(MW)	
11	Copper	0.18	0.27	3.6	3.8	3.3	+	
	Iron	18	25	210	230	210	+	
	Lead	0.08	0.12	1.5	1.9	2.0	+	
	Zinc	0.19	0.14	0.91	0.74	0.65	+	
12	Copper	0.17	0.50	1.6	2.9	3.9	+	
	Iron	20	60	160	210	280	+	
	Lead	0.12	0.38	0.86	1.2	1.3	+	
	Zinc	0.17	0.21	0.69	1.6	1.5	+	
Material		(CS)	(CS)	(CS)	(MW)	(MW)	(MW)	MW)
13[††]	Copper	*	*	0.20	0.15	2.0	3.4	*
	Iron	*	*	25	18	130	160	*
	Lead	*	*	0.12	0.09	1.2	1.7	*
	Zinc	*	*	0.21	0.16	0.88	1.02	*
14[††]	Copper	0.13	0.07	0.06	0.50	3.0	1.0	2.4
	Iron	26	17	14	40	230	210	140
	Lead	0.07	0.08	0.07	0.37	3.1	2.2	1.4
	Zinc	0.14	0.12	0.19	0.32	1.2	1.3	0.69
Material		(CS)	(CS)	(MW)	(MW)			
15	Copper	0.14	0.24	2.4	*			
	Iron	17	22	180	*			
	Lead	0.08	0.17	1.7	*			
	Zinc	0.15	0.18	0.96	*			

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[1] 0.31m is approximately equal to 1 foot.

+ No data available due to material compaction.

* Data not available due to sampling and/or analytical problems.

[††] The material in Columns 13 and 14 compacted approximately 0.31m (1 foot), therefore, data is reported here for only 2.2m (7 ft).

(continued)

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APPENDIX E (continued)
Soil and Solid Waste Metal Migration Data
(Dry Weight, mg/g)

Column Number	Metal	1st 0.31m[1] from top of column	2nd 0.31m from top of column	3rd 0.31m from top of column	4th 0.31m from top of column	5th 0.31m from top of column	6th 0.31m from top of column	7th 0.31m from top of column
16	Copper	0.06	0.08	*	2.5			
	Iron	16	17	*	*			
	Lead	0.08	0.08	*	0.51			
	Zinc	0.15	0.14	*	0.70			
17	Copper	0.04	0.28	3.6	1.9			
	Iron	13	28	180	*			
	Lead	0.05	0.81	0.48	1.3			
	Zinc	0.23	0.14	0.42	0.44			

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[1] 0.31m is approximately equal to 1 foot.

* Data not available due to sampling and/or analytical problems.

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